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Underground Industry

•

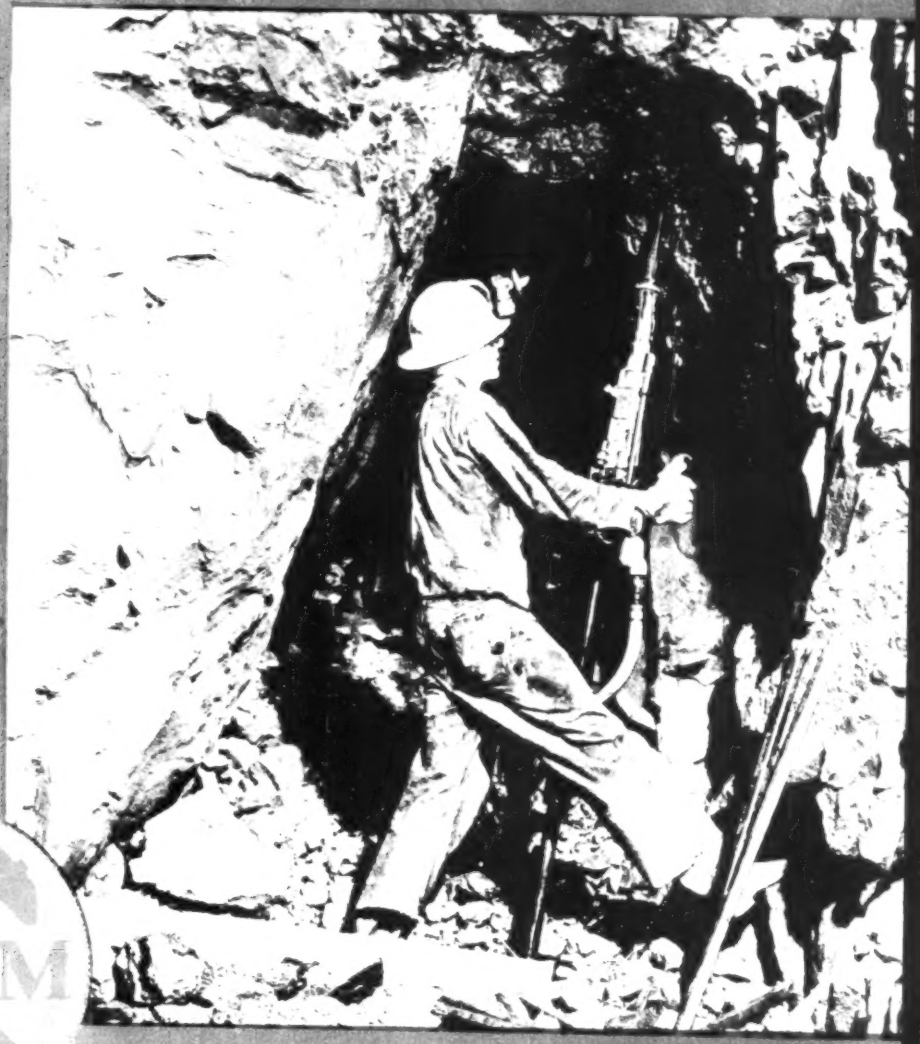
Countercurrent Decantation

•

Safety at the Crestmore Mine

•

AIME Officers and Committees



A MONTHLY PUBLICATION OF THE
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MINING BRANCH

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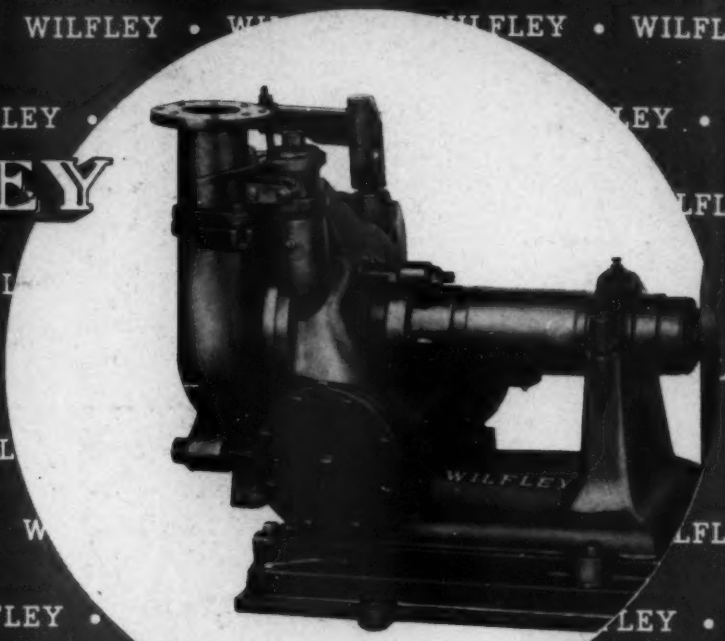

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CAST ROLL SHELLS OF THIS STEEL OFTEN INTERCHANGED WITH THOSE OF FORGED STEEL

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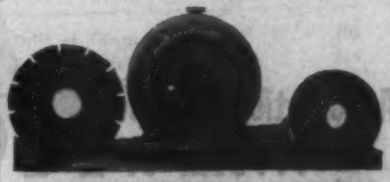
Taylor-Wharton's "Alloy No. 80" is utilized in castings such as this cement mill outlet grate, and for other applications where parts may be used rough-cast. They can be finished-machined where dimensional accuracy is required.



TO OBTAIN HIGH ABRASION-RESISTANCE

these sand pump parts were cast by the Taylor-Wharton Iron & Steel Company, High Bridge, N. J., in its "Alloy No. 80" containing on an average, 0.45% carbon, 2.00% nickel, 1.50% chromium and 0.35% molybdenum —

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of Alloyed Over Plain Carbon Steel Castings

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Safely and economically . . . cast nickel alloy steels with high carbon content provide sturdy resistance to abrasion, and withstand shocks, fatigue and various complex stresses encountered in crushing, grinding, and other processes of the mining, milling and smelting field. Typical applications include:

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- Grizzlies
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- Cams and tappets
- Crusher frames
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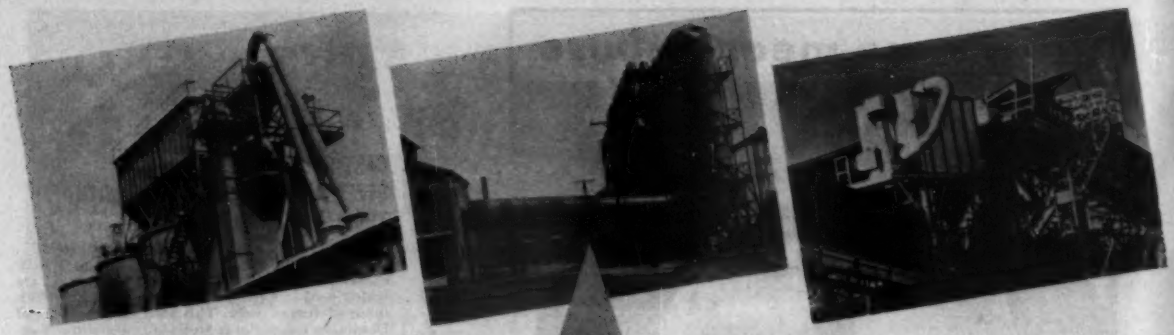

★ 1. With 78 cylindrical bags per compartment all the cloth area of each bag is in active use and bags are kept fully distended by air moving upward inside the bags.

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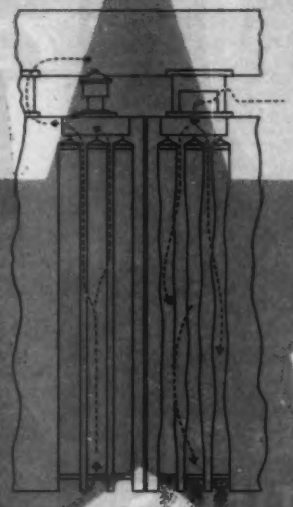

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MINING ENGINEERING

In which is incorporated Mining and Metallurgy, Mining Technology and Coal Technology

JUNE 1949 VOLUME 1, No. 6

Cover photograph shows blastholes being drilled with a stoper at the New Park mine near Kaslley, Utah. Story on p. 28.

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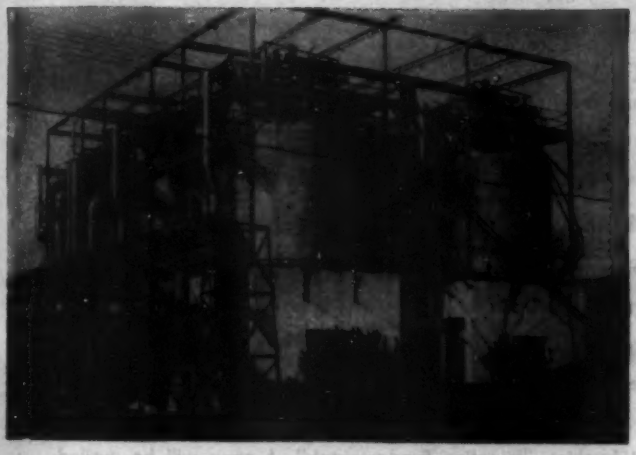
Mining operations are most successfully carried out if the water pumped into the sulphur deposit is heated under pressure to a temperature of about 320° F. For large scale mining, enormous quantities of water are required, so, a primary requisite is an adequate supply of suitable water and an efficient power plant in which to heat it.



To insure a continuous supply of water at Newgulf, it is the practice to use river water pumped in time of flood or full flow and stored in large reservoirs. This supply is supplemented, when necessary, with well water. Water so obtained is seldom suitable for use in boilers or mine water heaters without being treated first because of natural salts in solution. Softening by chemical treatment is necessary to prevent deposition of scale on boiler tubes and hot water lines.

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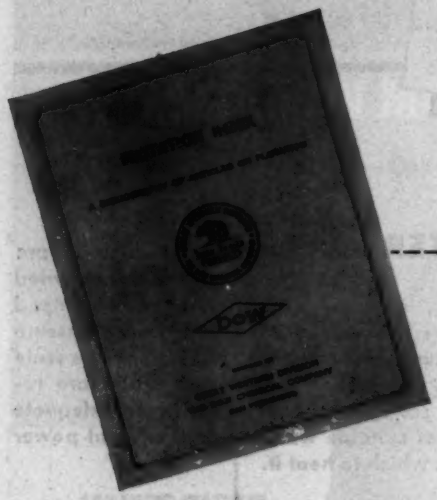

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MIE 6

Letter to the Editor

Recruiting for the Mining Industry

A distinguished engineer and educator comments on our April editorial "What does the Mining Industry Offer the Young Engineer?":

I do occasionally have to attempt an answer to the question your editorial poses. One can, of course, begin by pointing out, as this editorial does, that "This industry is essential to the American economy and the highest calibre men are needed."

One can go forward with some suggestions as to the great advances and operations in mining. The fact is this industry has been "tops" in its development of mechanization, in the power made available per worker, and in maintaining costs under the constantly increasing difficulties of handling lower-grade and more deeply-hidden ore bodies. These numerous developments are interesting and stimulating to young men of the engineering turn of mind.

But the profession as a whole does not seem to realize that we are in a new era. A fundamental change is in progress in engineering employment and, if mining wants to attract the best of today's students to the profession, the industry must revise its attitudes and methods.

In the old days the civil engineer, for example, started as a surveyor or draftsman. You will find today that it is rare indeed that a progressive industrial organization has any college graduates in its drafting rooms or on its surveying forces. Mighty few industries go to their drafting rooms when they seek men to promote to their engineering forces. The line between the technician and the college trained engineer is being ever more sharply defined. Engineering is constantly becoming more complicated, techniques more exactingly scientific and involved. Year by year fewer men can rise to professional status from the technician ranks. It is a basic characteristic of a profession that it becomes increasingly difficult for one to achieve professional stature through experience and self-study. Special instruction is essential and those who have it do not expect to start as draftsmen, or as workers in a mine. Industry hires both technicians *and* engineers.

Then there is another problem. The modern progressive industrial organization realizes that, if it wants to secure the best graduates of our engineering schools, it will have to go after them. You cannot expect them to come around knocking at the door, looking for a job. You may say: This is due to a shortage of engineers. It will be different when jobs become fewer. As a matter of fact, college recruiting, so-called, is not a new story, not a war baby. It has been a growing activity, stimulated undoubtedly by the tremendous postwar demand for engineers, but long recognized as the best way to get the best man.

When a student knows that he will be interviewed "on campus" by industrial representatives and will have several offers before he graduates if he is heading for the chemical, electrical, or manufacturing industries, whereas no mining companies ever send recruiting officers to the college and he will have to chase a job in mining for himself and will probably have to begin as a practical miner—well, unless his romantic make-up is strong, his interests will turn in other directions.

Our railroads are also "an industry essential to the American economy" but, with very few exceptions, they are in the same boat. Even our government engineering services, however, recognize the new situation and are taking steps to develop real recruiting programs and to enhance the opportunities of their staffs for continued professional growth and development. The college graduate is a college graduate because he is ambitious. He seeks that employment which gives greatest promise of satisfying these ambitions—and I do not mean purely financial rewards. He's a lot more mature in this respect and a lot more analytical and thoughtful in reaching a decision than many of the graduates of the good old days.

As one who has long had an interest in the mining profession I sincerely hope that the efforts of "Mining Engineering" to spread the new gospel will not end with the straightforward, realistic, and courageous editorial in your April issue.

JAMES KIP FINCH, Dean, School of Engineering, Columbia University

SEARCHING FOR BURIED TREASURE...

Flying Magnetometer speeds hunt for new oil, mineral resources in Africa

Long thought a treasure house of oil and mineral resources, Africa's new exploration and development flies ahead today with giant aerial strides. To speed industrial reconnaissance of the Dark Continent's great areas, American oil and mining interests rely on two precise mapping tools—the aerial camera and the newly developed airborne magnetometer.

Now the mapping plane explores *anywhere*, from the treacherous Canadian muskeg to the tsetse-ridden African jungle. In South Africa for example, the survey plane maps over terrain so rugged that ground parties would need months to cover the area flown in a single day. In Mozambique, the wild elephant herds, fierce rhinos and lions, as well as malaria and sleeping sickness, make ground surveys both slow and hazardous. *At a fraction of the cost of ground work*, the aerial survey collects engineering and geologic facts. And the airborne magnetometer probes the earth's crust for magnetic data—important guideposts for oil and mining geologists.

With 30 years' world-wide experience, Aero Service Corporation of Philadelphia is mapping a vast concession in Mozambique for an American oil company . . . also areas in the Union of South Africa for large mining interests. AERO will deliver mosaics and magnetic maps, as well as topographic maps for engineering planning.

Experienced AERO personnel bring important savings to mapping programs. AERO's record also includes many domestic mapping assignments for highways, city planning and tax maps. Ask AERO's engineers how aerial mapping can speed *your* work.

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236 E. Courtland St., Philadelphia 20, Pa.
Oldest Flying Corporation in the World

In the Union of South Africa, the airborne magnetometer flies over waste sludge piles of the gold mines. These tailings, sometimes as high as 200 feet of oozing mass, rarely solidify.

Barefooted, uneducated but friendly, these natives are their own beasts of burden.

Thorough pre-flight check of the high sensitivity airborne magnetometer.

Fierce elephant herds and other wild life are found within only 40 miles of Lourenco Marques, Mozambique's "Miami Beach."

Trek wagon and oxen, a typical South African trucking facility, bring gas to refuel the survey plane.

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"... from Little Acorns Grow"

Cyanamid Mineral Dressing Laboratory started 27 years ago as a humble one-room laboratory housed on a dock at the far end of our Warners, N. J. plant. It was founded — and its expansion has always been based — on the simple principle:

Cyanamid Reagents and Processes can never be better than the testing which precedes their application on your mill.

Today, the work of the Cyanamid Mineral Dressing Laboratory is just as important "A Cyanamid Product" as a drum of Aerofloat* Reagent or Aero Brand Cyanide. Most frequently, its findings come to you as a "Cyanamid Mineral Dressing Laboratory Report."

Just what is back of that Report?

Net, a Cyanamid Mineral Dressing Report summarizes the findings of scores of metallurgists, microscopists, physicists and testing engineers who in turn can draw upon the composite experience of nearly a thousand scientists in other departments of Cyanamid. Not even a small fraction of this technical talent and facilities could be economically justified by a single mining property.

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Obviously, the scope of the work going into each Cyanamid Mineral Dressing Laboratory Report differs with the specific problem. But, as the need warrants, a Report may combine the results of microscopical examination, exhaustive physical and chemical analyses; pilot plant testing by gravity, flotation, cyanidation, Heavy-Media Separation, Dutch State Mines Cyclone and various combinations of these methods. Conclusions are based on mature study and sound interpretation of the test data to evolve the most profitable flow scheme.

While ore samples tested each year number into the hundreds, they come principally from new ore bodies; from new deposits being opened on existing properties and from mills whose flow schemes must be modernized to stay economic. Frequently, however, Cyanamid Field Engineers can offer practical help right at the mill, based on previous results on an essentially parallel problem.

We invite correspondence and discussion with Cyanamid Field Engineers as to how the services of the Cyanamid Ore Dressing Laboratory can help you to increase yield, reduce cost or both.

**Aerofloat is a registered trademark of American Cyanamid Co. applied to certain flotation reagents of its manufacture.*

AMERICAN *Cyanamid* COMPANY

MINERAL DRESSING DIVISION

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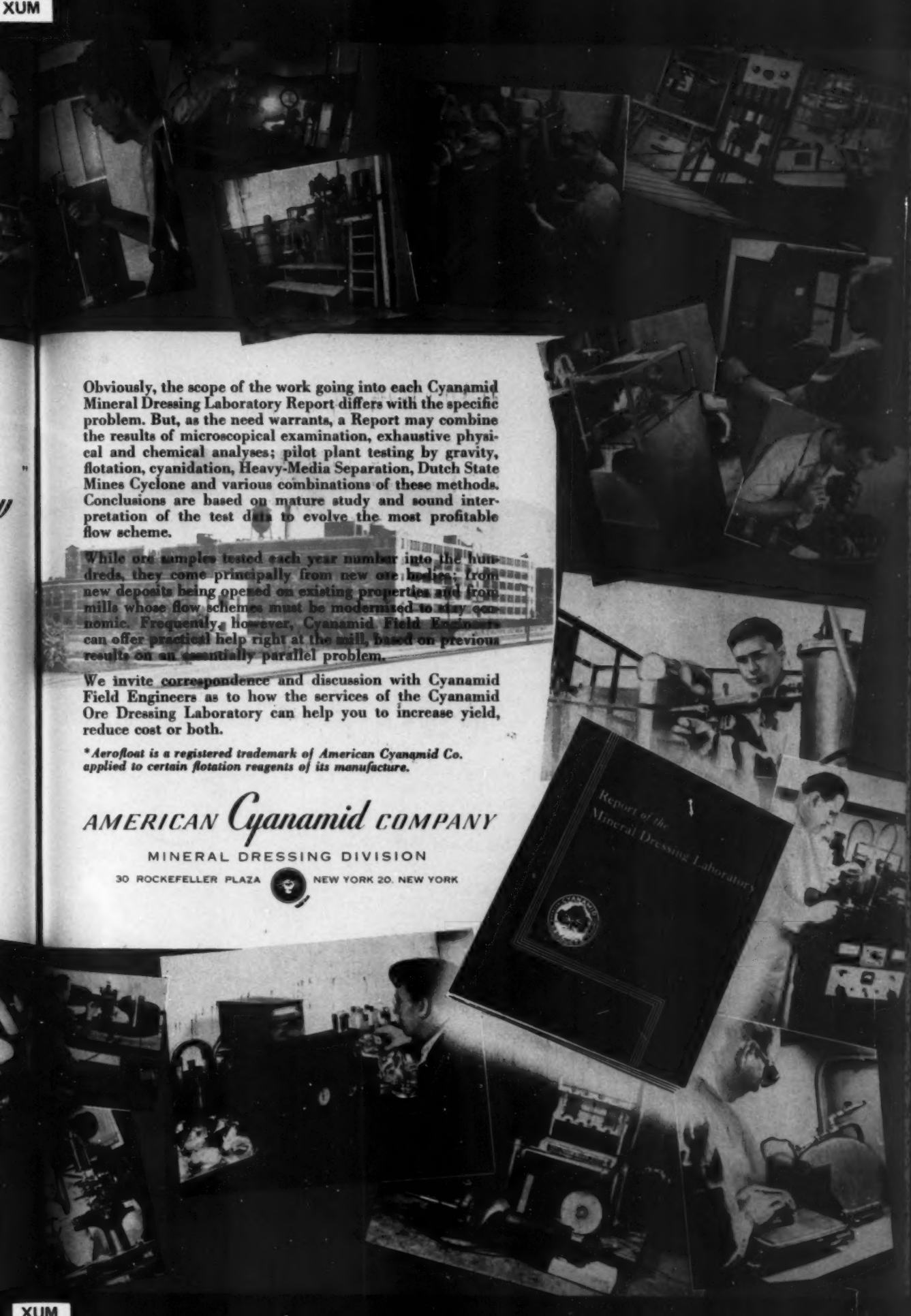

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Nine Years on the Job... Crushing Tough Trap Rock

INSTALLED IN 1940 by Bird and Sons, E. Walpole, Mass., this *Type R* gyratory crusher has been in operation 24 hours most days reducing hard, abrasive trap rock for roofing granules.

Trap rock is tough on crushers. Yet, so little "time out" for maintenance has been required since this machine was installed that these owners say it does a remarkable job for this kind of work.

In the modern *Type R* crusher, Allis-Chalmers has achieved both maximum strength and the elimination of excessive weight. It's built of sturdy steel construction with wearing parts made of heat-treated manganese steel. Fully automatic flood-lubrication of moving parts contribute to long operating life. Resilient mountings protect crusher and building from destructive vibration.

Type R crusher gives you maximum crushing capacity. That's due to the scientifically shaped crushing chamber... which also results in a more cubical product.

MAINTAINS EXACT PRODUCT SIZE

It's a topnotcher in operating convenience, too. "Speed-Set" control of crusher setting permits adjustment to exacting product size with turn of hand crank ... *easily*, without stopping crusher!

Automatic Reset lowers the entire crushing head hydraulically to pass tramp iron and other uncrushable materials... then brings it back to previous setting smoothly and without shock.

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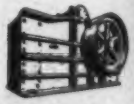
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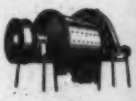
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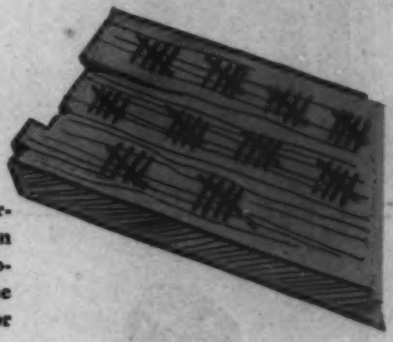

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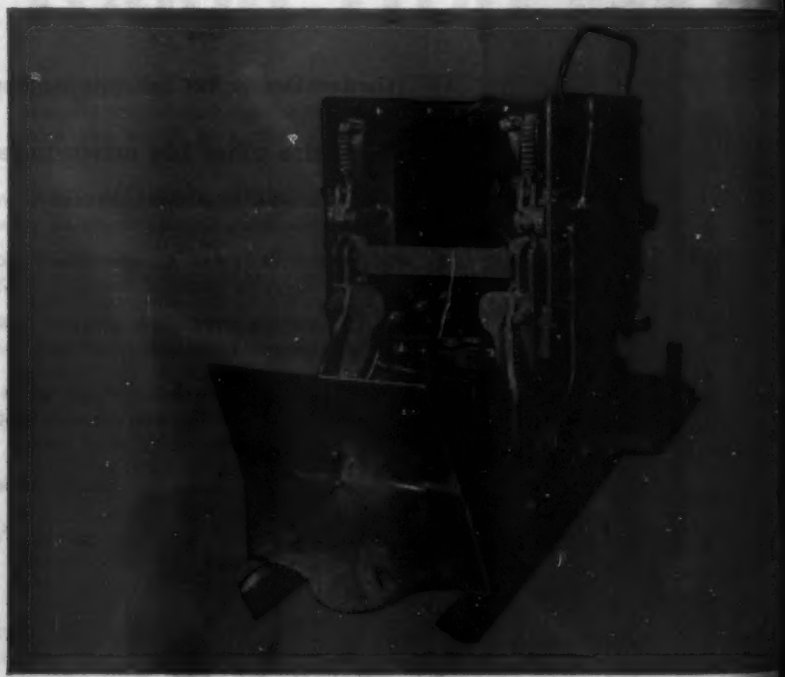

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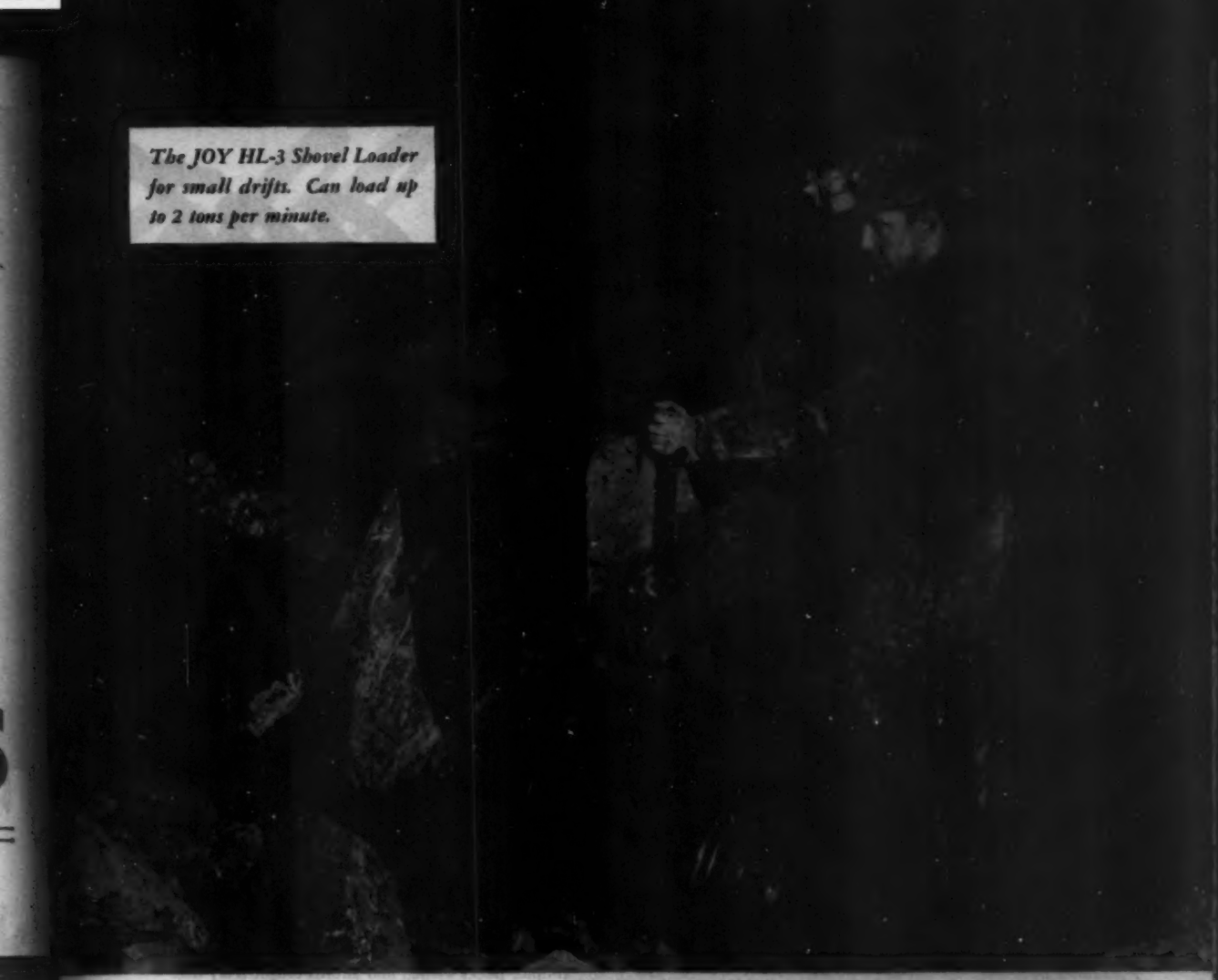

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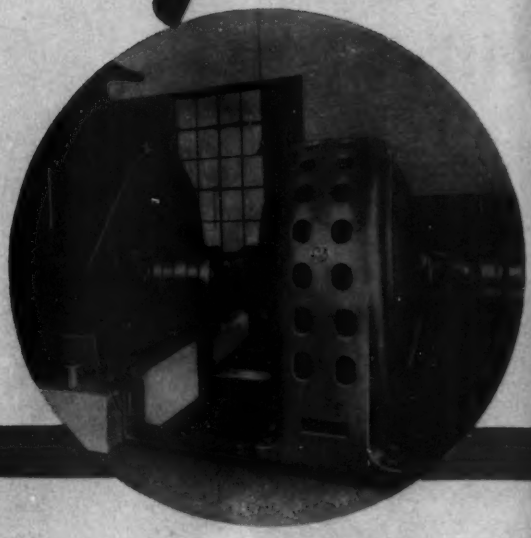
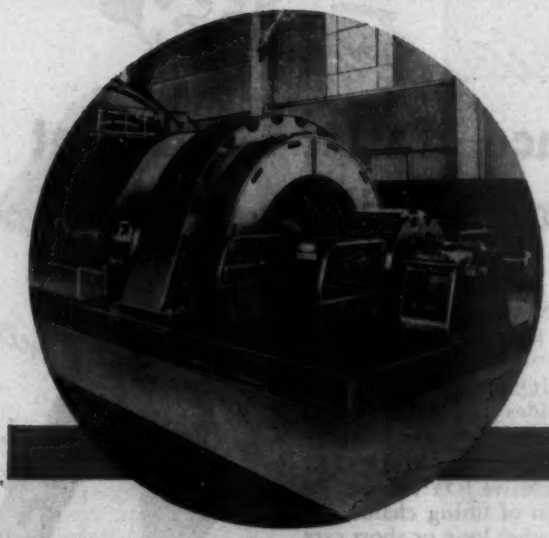
A typical G-E wound-rotor induction motor, 1000 hp, driving an ore hoist at a lead mine. G-E induction motors, ranging in capacities from 50 to 3500 hp, provide the simplest, least expensive method of obtaining the advantages of electric drive for mine hoists.



This 600-hp G-E induction motor is one of 16 identical units in a large Eastern mining operation. To meet specific duty cycles, G-E mine-hoist control uses the "definite-time" accelerating relay system that requires fewer interlocks, employs simpler panel wiring, and affords longer relay life.



A typical G-E 3500-hp wound-rotor induction motor geared to a main mine hoist. G-E motors and control equipment for mine hoists, designed and built for rugged service, have been on the job for many years in hundreds of mines throughout the country.

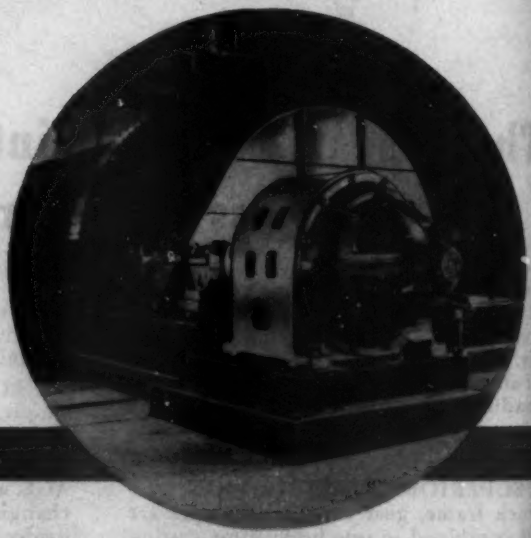


Photo shows a G-E 250-hp wound-rotor induction motor driving a man-and-material hoist in a Wyoming metal mine. G-E mine-hoist induction motors are low in upkeep cost, and require little or no maintenance attention. They need only routine lubrication and occasional brush renewal.

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with a G-E INDUCTION MOTOR Mine-Hoist Drive!

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In the G-E wound-rotor induction motors used for this service, the stator and rotor windings—specially insulated and securely braced—are ample for full "plugging" operation. All hoist-motor rotors are also strongly banded in place to withstand frequent reversing and above-synchronism operation.

SIMPLE CONTROL. Speed regulation is simple, and controlled through primary and secondary contactors by a master switch mounted at the operator's platform. Seven to ten successive speed steps are provided, from standstill to full-speed running position. Efficient, adjustable-speed control of overhauling loads is obtainable, where indicated, by dynamic braking, in which d-c excitation is applied to the motor stator windings.

G-E hoist control is designed to give maximum protection to machines and personnel in the event of abnormal operating conditions.

More than 700 performance-proved G-E drives—both a-c and d-c—are now helping mines throughout the country to step up production, and reduce hoisting costs. Whether you're installing a new hoist or modernizing an existing one, an experienced G-E mining specialist will gladly size up your problem, and help you meet it economically. Call him at your nearest G-E office. *Apparatus Dept., General Electric Co., Schenectady 5, N. Y.*

G-E enclosed mine-hoist primary control panel includes oil circuit breakers, air-break reversing contactors, line disconnects, and current and potential transformers.

Metal enclosures protect operating personnel, guard equipment against dirt and dust, and reduce maintenance and upkeep costs. Unit is factory-wired and assembled for quick, easy installation. Front view is shown above, and rear view at left.

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—to help the mining industry cut costs electrically!

GENERAL ELECTRIC

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MINING ENGINEERING

Supporting the American Way

One of the favorite topics for after-dinner speeches and editorials these days is the encroachment of government on industry which some think presages the approach of the socialization of our economy. This subject is vital to professional men in the mining industry, for many of them ultimately become the leaders in this field. The mining industry was founded on the system of private enterprise, and individual initiative was the mainspring that drove the system. Thus it is that the very nature of our business, our education, and our upbringing make the new trend odious to us.

Industry has the answer to this threat and, if appropriate action is taken immediately, the march toward socialization can be halted.

We propose a three-point program that can be started right now. (1) Educate the public to the advantages of free enterprise. (2) Put into practice a real program of intelligent management that recognizes that production can best be achieved by the co-operation of capital, management, and labor, each receiving an equitable return for their investment, be it money or work. (3) Utilize engineering methods to cut operating costs and develop engineers for executive posts by training them in labor relations, economics, and public relations, as well as in operation.

It is not enough that the public enjoys the fruits of the American way in the form of material advantages; the public must have these things pointed out and be told where and how they became available. We have been too prone merely to discuss these things among ourselves, among those whose ideas run parallel to ours. It is time to take our arguments to the public through every available channel.

In a recent address before the Executive Club of Chicago, Harvey S. Firestone, Jr., recommended that channels for disseminating the superiority of competitive free enterprise could be found in addressing our employes, our children, Boy Scout Troops, YMCA members, parent-teacher groups, women's clubs, and the like. He set down the realistic advantages of the system which are so widely taken for granted.

In speaking of our superior standard of living, Mr. Firestone nails it down with a few facts:

(1) With only 7 percent of the world's population, we have a higher degree of culture evidenced by our having more young people in high schools and colleges, more musical and literary organizations, more libraries and more distribution of the written and spoken word than all of the other ninety percent combined.

(2) We hold a position of moral and spiritual leadership exemplified by more hospitals, charitable institutions, and probably more churches

per capita than any other country. Although the conqueror in two world wars we have aided friend and foe alike to rehabilitation.

(3) For a comparison of material things between the United States, Great Britain, and Russia a single example is typical. There is one automobile for every four citizens in the United States, one for every 22 in Great Britain, and one for every 252 Russians.

(4) Our system is tried and proved successful whereas other systems "are either untried experiments or downright failures."

(5) Let us correct the impression that the owners get fabulous profits on every sales' dollar whereas in reality it is more like three cents; 85 cents goes for salaries and wages.

(6) "Let us tell the real facts about who owns American business . . . Everyone of the 50 million Americans who has a savings account, each of the 54 million who has a life insurance policy, and each of the 14 million who has shares of stock or industrial bonds, owns directly or indirectly a part of American industry . . . A national income study made by the Bureau of Economic Research of the University of Notre Dame, shows that 90 percent of all wages and salaries, nearly 70 percent of dividends and interest, and 83 percent of income from rents were paid to those earning under $5000 a year." This same study shows that, after payment of Federal taxes, only 5½ percent of wages and salaries, 17 percent of all dividends and interest, and 9½ percent of income from rents goes to those earning more than $5000 a year.

These are the facts which should be disseminated to the public.

In addition to spreading the gospel of free enterprise, progressive management must be fostered. While the public takes for granted the produce of our economy, industry may be guilty of forgetting that it takes the co-operation of capital, management, and labor to produce goods. Each is entitled to a fair return on his investment. Labor unions have sprung from the abuses of management against labor. Most of these abuses have been abolished but unions will disparage management as long as their leaders can find a crevasse of decay in which to drive a wedge. By thinking ahead and forseeing needed adjustments industry can inspire the confidence of labor and a condition of harmony and prosperity can mutually be realized. The workers must be taught that they have a vital interest in the prosperity of the industry in which they are employed. If co-operation with management nets them the greatest return for their invested labor, it won't take much selling to convince them of the efficacy of the existing system.

Another phase of a comprehensive program to entrench the American way against the rising tide applies particularly to the mining industry. More engineering principles must be applied in our operations and engineers must be broadly trained in all phases of company organization to ensure a supply of capable executives. Both large and small marginal properties need superior engineering but frequently this is where engineers are considered expendable overhead. Some organizations don't use their engineers to improve methods but permit them to spend several years on routine record work while waiting for a vacancy in a supervisory position. Frequently an engineer who has shown good technical ability at the mine over a period of years is transferred to the office and put to executive work without any preliminary exposure to taxation, marketing, or policymaking problems. Engineers should be used for studying, planning, and improving operations, and constantly exposed to work requiring imagination and the ability to master a new phase of company business. Such a procedure will improve company operation and insure a supply of properly trained men to fill executive positions. They will have the background to ensure progressive management.

Educating the public will relieve pressure on our legislators who may otherwise feel they must back paternalistic or industry-control laws to satisfy their constituents. Progressive management can remove the conditions that make regulatory legislation necessary. Finally, greater use of engineers and management training will pay the way for all with greater profits, making ore out of waste. Let each of us examine his station in life and see what part we can play in protecting our birthright of freedom.

Industry Contributes to Education

Recently we had the pleasure of accompanying a group of young men and women from the Department of Geology and the School of Mines of Columbia University on a field trip through Pennsylvania. We went through the Philadelphia & Reading Co.'s coal breakers, mines, slate quarries, and mills; the Nazareth Cement Co.'s pit and plant; and the Bethlehem Steel Co.'s plant, Cornwall Mines, and mill. The pace was rapid, and we saw much in only three days.

The value of such trips to both students and industry is unmistakable. For the student, classroom work comes to life and takes on meaning. Paper work becomes the pulse of machines, men, and raw materials blending together to produce a commodity.

Now industry may find such hospitality expensive, especially when it provides guides, in-plant transportation, and brochures or guide books for students, or the general public. To such co-operative organizations, we offer a vote of thanks. We are sure they'll find, however, that the seeds of sociality grow good crops. Many a young graduate has returned to a place where he was properly impressed, and become a worthy member of the organization. Then, since the colleges are training students in engineering and geology for positions in industry, it is to our benefit to see that they learn something about what their future careers will be like, and these visits provide a welcome glimpse into the years following commencement day.

Some companies also find it advantageous to hold open house for their employes' families and members of the local community, and thus develop good will and a broader community understanding of company problems.

Take a second look when those eager students descend on you. They'd all like to be president of your firm. And one of them may make the grade.

This Tudor-type arch was achieved by careful blasting at the Bolinder-Munktell underground engine factory near Eskilstuna, Sweden. Low maintenance counteracts initial high expense.

Photos courtesy of Aviation Week

Machine tools can be arranged as efficiently underground as above surface, as shown here. Proper use of color gives a feeling of roominess.

Underground Space for American Industry

a new job for the mining engineer, geologist

By GEORGE A. KIERSCH • CORPS OF ENGINEERS, SACRAMENTO, CALIF; JUNIOR MEMBER, AIME

The awesome destructive power of known and projected weapons of war presages a new need for geologists and engineers, who may be called upon to locate vital industry underground, thereby protecting it from sky-borne destruction. The Corps of Engineers is now conducting special tests[1] to establish design standards covering the general field of underground installations as well as surveys[2] of construction methods and the operational hazards connected with such plants.

National planning must dictate a policy that is either tactical, where surface manufactured parts are stored underground and during an emergency assembled, or operational with both the manufacturing and assemblage completed in subsurface plants. Both plans of operation require the use of subterranean plants. Re-location of vital plants underground will probably involve only a small percentage of the production capacity of this nation. The cost of rebuilding in new locations will obviously limit its adoption.

Before these plants are built, research will be necessary to further determine the effects of explosion on rock masses and the type and size openings that can be constructed within these rock masses to withstand the dynamic stresses that might be imposed on them. Construction techniques will also have to be studied so that these subterranean plants can be built rapidly and at minimum cost.

A study of abandoned mines indicated that those in limestone, gypsum, and sandstone were the most desirable for subterranean plants, and that several hundred million square feet of usable space was available in such mines.[3] However, except for these nonmetallic mines, other existing openings may be unfit for subterranean installations due to: location in structurally weak rock or alteration areas; not designed for maximum stress resistance; cross sectional outline too large or too small; shaft entrance only; water table too high; plan of workings will not conform to an assembly line layout.

Some caves may be found to meet factory requirements, but the probability of much remodeling is most likely. Commonly, caves do not have a uniform floor level, rooms are irregularly aligned, and have a high humidity. It is probable that an entirely new system of openings completed by highly mechanized methods would compare favorably in over-all cost to an expensive job of modifying an abandoned mine or cave.

rock structure important

Some of the major engineering and geologic factors necessary for a successful underground construction program include: (1) site reconnaissance, exploration, and interpretation of the geologic conditions; (2) plant design and construction; (3) machinery installation and maintenance of these subterranean plants.

Safe resistance to combined static and dynamic stresses is an essential

Views expressed herein are those of the author and not necessarily those of the Corps of Engineers.

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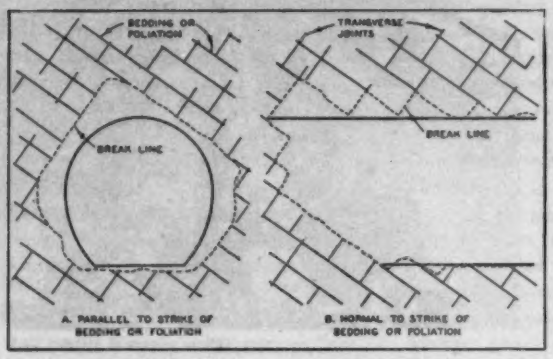

BEDDING OR FOLIATION
BREAK LINE
A. PARALLEL TO STRIKE OF BEDDING OR FOLIATION
TRANSVERSE JOINTS
BREAK LINE
B. NORMAL TO STRIKE OF BEDDING OR FOLIATION

Fig. 1. Horseshoe cross-section in moderately-steeply dipping beds or foliated rock breaks to undesirable dashed outline.

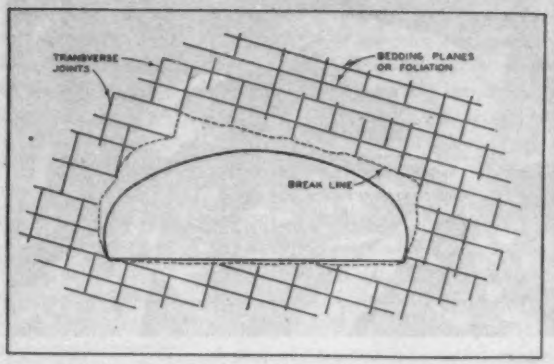


Fig. 2. Tudor-type arch in gently dipping beds or foliated rock will be impractical. Note line of break.

property of any proposed underground plant site. Burden measured in hundreds of feet will probably be required in some rock types to protect adequately personnel within a large underground structure. Factors controlling the depth of burden include: physical properties and inherent geologic conditions of the site rock, depth of penetration for the center of the explosion, and size and shape of the subsurface opening. A smaller opening will mean less static pressure on the walls,[3] and increase the amount of dynamic stress the structure will be able to withstand. Knowledge of the sizes and shapes of unsupported openings commonly used in mining and tunneling will be of some value, but it must be remembered that few of them have been subjected to other than statically imposed stresses. Studies such as those conducted by Wright and Bucky[4] can provide data on the ability of each rock type to stand unsupported under static loading, and aid in the design of structures able to withstand a superimposed dynamic stress. C. W. Livingston, of the Colorado School of Mines, has designed a series of small-scale field experiments[5] that can be helpful in estimating the type and severity of damage to be expected from large explosions in rock. These data will be valuable in comparing the properties of an unknown site rock with that of previously tested and studied rock masses. Such investigations will be necessary for designing the underground plant and aid in the selection of a tunneling method.

The size and location of underground plants, as well as the type of site rock and tunneling methods employed will be dictated by the required plant layouts and the size of the machines to be installed. Areal limitations on location may require changes in the original design to meet local geologic conditions.

Subterranean installations should avoid areas characterized by an abundance of structural deformation and chemical rock defects. Jointing, the most common rock defect, may be so closely spaced and extensive as to have major structural importance. The primary joint set will probably cause less trouble if the centerline of the opening is laid out normal instead of parallel to the strike of these breaks. Sites in moderately to steeply dipping bedded or foliated rock will overbreak during excavation and slump on the up-dip wall (Figs. 1 and 2) and are thereby unsuited to certain designs.

A thorough geologic exploration program is necessary prior to final site selection. An aerial photo survey, which can serve to reveal important rock defects, can be followed up by an adequate drilling program to prove-up the site and locate otherwise unsuspected geologic conditions. The location of the water table above the proposed floor level, and the consequent necessity of redesigning construction plans, is one of the many obstacles inherent, and discoverable, in underground construction. Certain geophysical methods of exploration can be used to determine boundaries between rock types and zones of structural

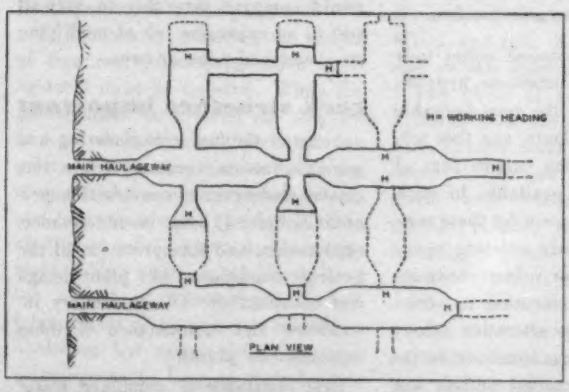


Fig. 3. Multiple headings, here shown, are adapted to modified room-and-pillar method of tunneling.

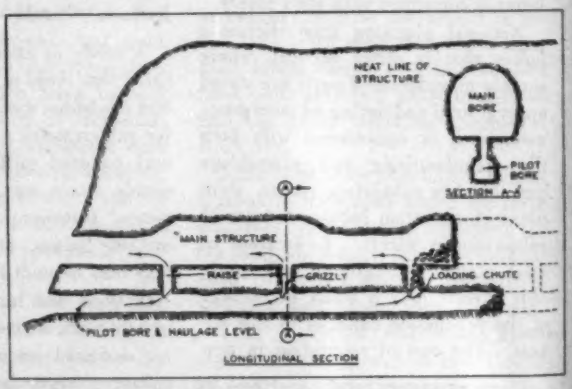


Fig. 4. Easy rock handling is achieved by haulage below main structure and connected by raises. Arrows indicate air flow.

weakness. In addition, horizontal drilling and pilot bores will serve to fill-in geologic information lacking from surface observations and drill core surveys.

Subterranean plant design must combine the technical skills of mining, tunneling, construction, and geology with the principles established by past and future research in the field of protective construction.

The Germans employed two types of plants.[6] The semi-underground bunker, a solid reinforced concrete span, was used where tunneling was impossible. Subterranean plants located in abandoned mines and quarries revealed nothing new in design to the American engineers.[7]

In Sweden, where several underground plants are now in operation, the Tudor-type arch (Fig. 2) has been adopted for cross-sectional design in granitic rock. The opening is wide, yet reasonably low. In England,[8] a subterranean plant was located in horizontal chalk beds 150 ft below the surface. Underground structures of a modified-arch shape, 22 ft wide by 20 ft high, were excavated. Steel supports and concrete lining were required in fractured areas.

Mining and tunneling practice has demonstrated that a circular tunnel will withstand greater static rock pressures than any other shape cross section of equal area.[9] The horseshoe cross section (Fig. 1) is a widespread modification which withstands greater static rock pressures than the more common flat or arched outline. Use of these common tunneling cross sections will depend on the character of the site rock and the size of the structure. For example, a site in a tilted bedded rock mass would require a different design from one in a horizontally bedded rock of the same type.

mechanized tunneling needed

Large-scale underground construction will require the use of several types of tunneling methods, drilling, loading, and haulage equipment, and ventilation layouts and machinery. Such a protective construction program probably could be initiated during an emergency and would therefore require complete mechanization to attain speed and over-all efficiency. The tunneling method used will depend largely on the character of each particular site rock and the plant requirements for size and layout. The term "tunneling methods" refers to all types of subsurface excavation connected with the program.

A valuable modification of the room-and-pillar mining method is that of driving two or more main bores parallel and a specified distance apart by the fastest method. Simultaneously, or later, larger structures are located between these bores and connected by tunnels as in Fig. 3. The main bores would serve as haulageways. Specialized tunneling crews could move rapidly to the multiple headings and perform their part in the operation cycle, thereby increasing both the personnel and operational efficiency. In this manner many small to medium-sized structures are possible; tunneling operations may be advanced in several separate headings simultaneously off the main haulageway; waste rock is easily transported to main haulageway for rapid removal; and ventilation is easy because of natural circulation through the network of structures.

pilot bores speed work

Ahead of excavation of the main structure and at a safe distance below its floor a pilot bore is driven as in Fig. 4. As the main heading progresses, connecting raises are driven from the pilot bore behind the advancing face of the main structure. Waste rock is scraped, dozed, or otherwise mechanically handled to the raises which are equipped as rock chutes; the pilot bore would operate as a haulage level. This method has the following advantages: no rock haulage necessary in main bore; quick clean-up after blast; highly mechanized and thus speedy operation cycle; natural ventilation during construction, with utilization of the pilot bore for air circulation during the factory operation; pilot bore could serve as the main haulageway during factory operation with lifts to the plant floor installed in the raises. This method is adaptable to a plan calling for either a single main bore or for one having parallel bores with structures between as in Fig. 3.

Another method, shown in Fig. 5, would use pilot bores along floor and roof with connecting raises. This method would be applicable where a subterranean structure of large cross section was planned. Pilot bores are driven as fast as possible along the floor and roof of the proposed sub-

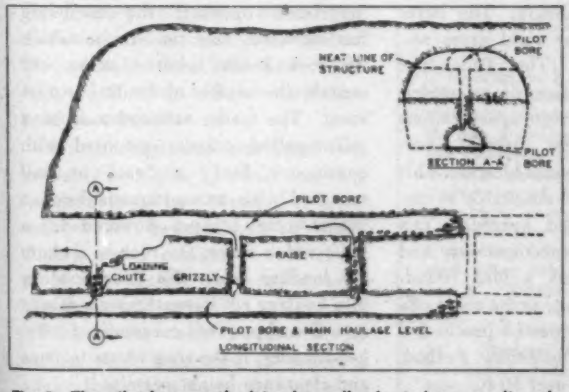


Fig. 5. The pilot-bore method is also well adapted to speeding the excavation of large cross-section openings.

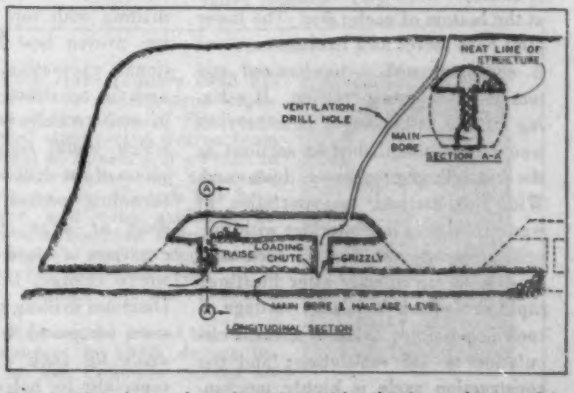
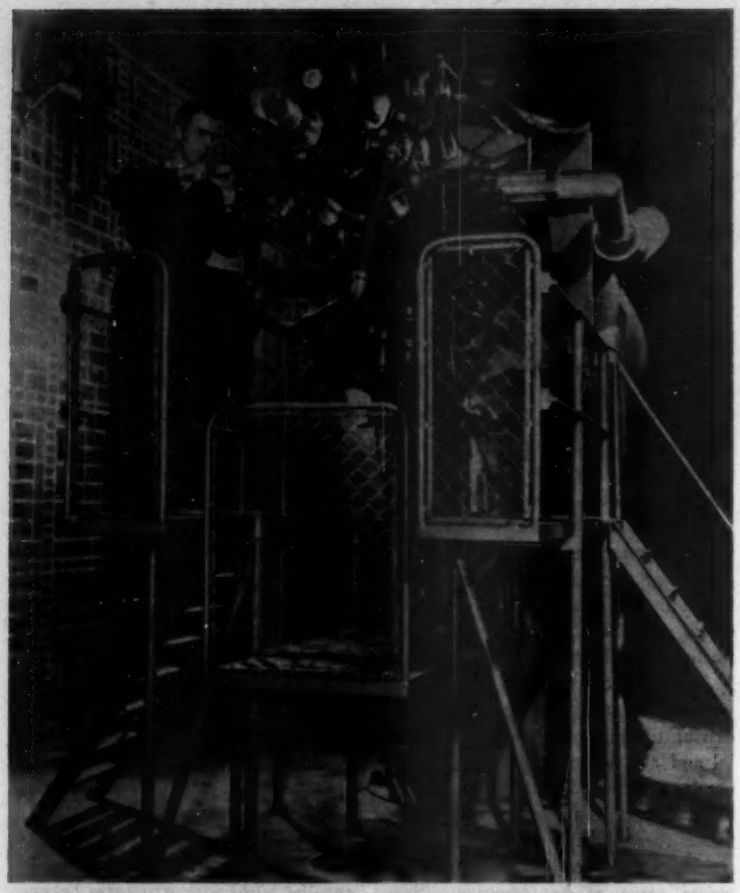


Fig. 6. Main bore-enlarged sections with pilot bore along roof is similar to method employed by British in tunneling in chalk.



An engine testing room like this will be soundproofed by the natural rock pillars. Ventilation ducts can be drilled through the overlying strata, zig-zagged to eliminate bomb damage; lead seals in all outlets could stop radioactivity of atomic bombs.

surface structure and are connected at specified intervals by raises. Vertical blastholes can be drilled by wagon drills or other appropriate machines from the upper bore and, after widening to cross sectional size, the broken rock scraped or dozed into the raises. A grizzly with draw chute is installed at the bottom of each raise. The lower pilot bore serves as a haulageway, and is equipped with a mechanized system for rock transportation. If a lining of the structure is necessary, it would be accomplished in sections as the tunneling progresses downward. With this method, transportation of waste rock does not interfere with tunneling operations; broken rock may be cleaned up quickly after blasting; rapid excavation of a large yardage of rock is possible; there is natural circulation to aid ventilation; and the construction cycle is highly mechanized. This method should not be considered where the structure cross section is small or moderately sized.

enlarging initial bores

An initial pilot bore, driven at maximum speed, can be enlarged to the specified structure outline by long hole ring drilling or by horizontal drilling with top slicing. The latter has proven best for rigid cross sectional excavation. The Tudor-type arch is an illustration of an underground structure of rigid specifications which would require careful supervision of the drilling and blasting. This tunneling method is adaptable to medium or large sized openings, can excavate to rigid specifications, and where desired, break a long round. Diamond drilling may prove more efficient compared to jointed percussion drills for such a tunneling method, especially for holes over 50 ft.

A main bore driven along the floor is enlarged to a big structure size in certain desired sections as in Fig. 6. Within these sections, after raising or enlarging, another small tunnel is driven along the roof of the proposed structure which subsequently is widened and shaped to specifications. The rock between the two bores is removed by vertical blasthole drilling; waste rock is scraped to the raises or enlarged sections now used as chutes; mechanical loading and haulage can be used in the main bore as shown in Fig. 5. This method is especially good for excavation of large-sized structures several hundred feet long which are connected by smaller tunnels to others of a similar size. The British[8] used a somewhat similar method in their tunneling in chalk.

new equipment problems raised

To speed construction, improvements will be needed in the present drilling, loading, and haulage equipment. Using present drilling equipment, a jumbo mount on rails, Diesel powered or air driven "cat" treads would be best. Long sashes on the drill machines permit longer holes without changing drill rods. A long sash feed mounted on a hydraulic drill jib further saves time and work in moving the machine to complete a round. Drill bit changing can be cut to a minimum where the longer-life tungsten-carbide bit is adaptable. For long hole drilling, a jumbo mount for air-driven diamond drill machines may be very effective. The continuous mining machines will probably find wide use in softer rock types.

The cross sectional size of the underground opening, the tunneling method used, and the size to which the rock breaks after blasting will control the choice of loading equipment. The loader selected may be a selfpropelled crawler mounted with continuous feed; a "cat" or rail mounted with noncontinuous feed; a conventional scraper powered by a hoist which moves the rock to a chute for loading below the floor level or into haulage equipment using a ramp; or a Diesel-powered cat-mounted overhead loader, if the rock blasts to fines and clearance height permits.

Both the tunneling method and the

cross sectional size of the opening will have a bearing on the type of haulage. Conveyor belts, widely adopted for underground transportation of certain type rock, would be especially adaptable for use in a pilot bore or a main haulage level. Fast, trackless shuttle cars are easy to load and unload and operate in low heights. The average type of truck used for such haulage would require a larger opening for operation. Track haulage with mine cars could be used if tunneling operations required the laying of steel track.

Another aisle in Bolinder-Munktell plant. The even temperatures of underground plants reduce heating costs and, with proper air conditioning, greatly improve workers' health. Fake windows backed by outdoor scenes can be used to combat closed-in feeling.

new problems are challenging

A wealth of new techniques, and ingenious improvement on old ones, are called for in this new field of protective construction. Training for the work will call for a combination of mining, civil engineering, and geology, enabling young graduates to step directly into the new industry as it develops. The Colorado School of Mines is presently providing to Corps of Engineer officers special training in mining engineering principles and accenting underground construction.[10] Three officers have already received their Master of Mining Engineering degrees at the school.[5]

Arrayed side by side with the basic underground reinforcement and ventilation problems known to the mining industry today, are the myriad complexities ranging from extensive camouflaging of excavated material when secrecy so dictates, to the subjects of ventilation, lighting, and even prevention of worker claustrophobia! Pilot bores and connecting raises as in Fig. 4, 5, and 6, and drill holes as in Fig. 6 can provide some natural ventilation—but machines and men and production methods will have to be adjusted to many new, seemingly unnatural, conditions.

New underground structures will probably be higher in initial cost than comparable surface plants, but lower maintenance, heating, and air conditioning costs, combined with cheap protection against destruction, must also be considered.

Pilot plants could be erected by certain vital industries, in order to adapt their manufacturing cycles to the new conditions. Potential site locations include the limestone formations of the Ohio and Mississippi valleys, interbedded sandstone shale, and limestone formations in the intermountain areas, the Pacific Northwest's plateau basalts, the granitic rocks of the Sierra Nevada, Idaho, and coast range batholiths of the West.

The Corps of Engineers has been assigned the responsibility for acquiring, developing, and disseminating within the military establishment, protective design criteria and for designing and constructing underground installations. Data are being developed for application to underground construction should it become necessary in time of emergency.

Many thanks are due the following for their suggestions and stimulating discussions: H. E. McGee, J. C. Coombs, P. M. Johnston, and C. J. Gorman, of the Sacramento District, Corps of Engineers; A. S. Kane and Robert Selby, of the Morrison-Knudsen Construction Co.; C. W. Livingston, Colorado School of Mines, and M. N. Short, University of Arizona, in addition, criticized the original manuscript. W. Ralph Forssell, Sacramento, has drafted the illustrations.

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[10]*Mines Magazine*. Nov. 1947, p. 74.

George A. Kiersch since January 1948 has been associated with the underground explosion test program of the Corps of Engineers. He has a geological engineering degree from the Colorado School of Mines, and a Ph.D. in mining geology and engineering from the University of Arizona. Dr. Kiersch has been a miner, surveyor, and mine geologist. He was, for two years, a teaching fellow at the University of Arizona's geology department, and after taking his doctorate, went on to teach economic geology at the Montana School of Mines.

Now a Junior Member, he joined the AIME as a Student Associate in 1938, during his freshman year at the Colorado School of Mines.

Four Euclid trucks and 1 to 2½-cu yd shovels were used to clean up, handling about 1000 cu yd per 10-hr shift.

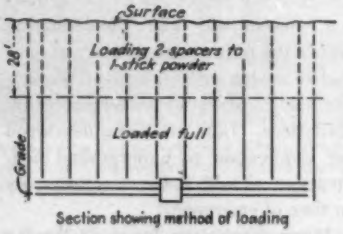
The clean, square bottom of this cut, shown strikingly in this photo, was obtained by using diamond-drill blastholes.

Diamond Drills Excavate Channel

By **CHARLES HOPPER** • CONSULTING ENGINEER, BOYLES BROTHERS

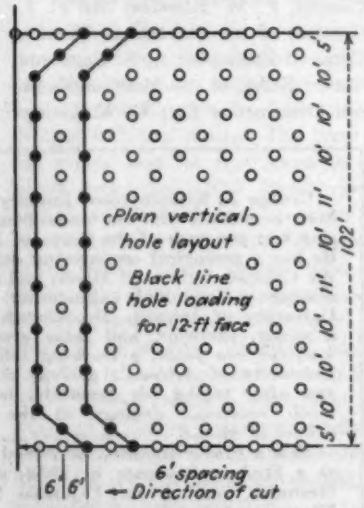
In preparing the Steep Rock Lake iron ore body for mining, it was necessary to drain Steep Rock Lake. Using diamond drills, a cut 1800 ft long, 100 ft wide, and maximum depth of 95 ft amounting to 300,000 cu yd of material, was excavated in Greenstone to provide a drainage channel. This was done under the general contractor C. A. Pitts, of Toronto, who in turn employed a diamond drill contractor, Boyles Bros., to do the drilling. The method of excavation was selected because a clean square cut and grade could be maintained in the bottom of the channel. Bootlegged holes would have left blocks of rock in the corners which would have created a situation which would have become progressively worse as the open cut advanced from one side of the ridge to the other.

A 7 by 7-ft tunnel was driven along the center line of the proposed cut for its entire length. Rows of three horizontal diamond-drill holes, 45-ft long, were drilled on either side of the drift, the rows being on 3-ft spacing. In order that the bottom holes might be on grade of cut, it was necessary to drive the tunnel three feet below grade so that drill setups could be made. The vertical holes were also drilled with diamond drills from wooden staging, three holes being drilled from each setup and spaced as shown in the diagram. Two rows of vertical holes were blasted for a 12-ft advance. In each case, the outside hole was left till the following blast, as the corner would not come out otherwise. The lifter holes were loaded solid, and vertical holes loaded solid to within 28 ft of the surface, where two spacers to one stick of powder were used. Both horizontal and vertical holes were blasted simultaneously using 75 per cent Forcite 6 by 1¾-in. sticks and instantaneous caps and primacord.



Section showing method of loading

Bottom horizontal holes from tunnel are driven on grade.



The vertical holes were also drilled with diamond drills.

Each 12-ft face advance averaged 3000 cu yd of rock which required from 26 to 36 hours for 1 to 2½ yd shovels and four Euclid trucks to clean up. The average amount of rock removed per 10-hr shift was about 1000 cu yd. Lifter holes did not seem to have sufficient lift to clear muck away from the face, which slowed shovels up. If the horizontal lifter holes had been drilled out one foot into the wall and vertical holes drilled three feet below grade on the outside row, the corners would probably have been cleared. An additional lifter hole would also have helped.

The average footage drilled per 10-

Water from Steep Rock Lake was drained through this channel in order to mine the iron ore underlying the lake.

hr man-shift, including timbermen and all other men on the job, was 45 ft. The average footage drilled per machine drill-shift was 120 ft. Powder consumption per cu yd of rock broken was 0.6 lb.

1000 at Annual Canadian Institute Meeting in Montreal

Reported by E. H. Robie

Approximately one quarter of the total membership of the Canadian Institute of Mining and Metallurgy, which with students numbers about 4000, attended the 1949 Annual General Meeting in Montreal, April 25-27. In addition, more than 250 ladies were registered, so that the meeting was one of the largest ever held. Headquarters were at the Windsor Hotel, with sleeping and meeting room accommodations conveniently supplemented by the new Hotel Laurentien just across the street.

Montreal's well-known mayor, Camillien Houde, welcomed the visitors at the Monday luncheon most affably. Perhaps "intriguing" is as good an adjective as any to describe his presence. His leadership of the French Canadians in the war landed him in a concentration camp, out of which he no doubt beamed his way. He was definitely an attraction at the luncheon. Other welcomers were C. D. French, the Provincial Minister of Mines, who put in a plug for New Quebec as a mineral storehouse of the future, with its newly developed iron and titanium, to say nothing of Shipshaw and the other waterpower developments. Ralph W. Diamond, retiring president of the CIM, added a few words of greeting and then our own L. E. Young made a fitting response.

One of the three papers on the general session program in the afternoon was by Edward Sampson, professor of geology at Princeton. The program concluded with an exceptionally well-made sound Technicolor film of the operations of the Consolidated Mining and Smelting Co. of Canada—"No Man Is an Island," the theme of which is teamwork.

A cabaret dinner for ladies and gentlemen concluded the day with the best in food and entertainment.

Four of the five Divisions of the CIM—Geology, Coal, Industrial Minerals, and Metallurgy—had separate programs Tuesday morning. The geologists and the metallurgists again had sessions Tuesday afternoon and Wednesday morning, and the Metal Mining Division had one session Wednesday morning. A concluding general session was held Wednesday afternoon, and the incoming and outgoing councils held a meeting on Thursday. Many papers of considerable interest were offered.

Donald H. McLaughlin gave a paper on the Homestake mine, and included a bit of argument for better treatment of gold producers. Gold bulks large in the mining economy of the Dominion, and the gold producers up there have not been very well satisfied with their lot, though they were never completely closed down by government ukase during the war, as they were on this side of the line.

There was much talk at the meeting of the big new oil finds in Alberta, which promise to be comparable with the biggest discoveries in the U.S.A., and also of the iron and titanium in Quebec and Labrador already mentioned. More uranium, to supplement that afforded by Eldorado at Great Slave Lake, has also been found, and all in all mining development in Canada is on the up and up. Metal production was valued at $484,000,000 last year, an increase of 22 percent, and mineral fuels at $159,000,000, up 44 percent.

Canadian Institute meetings run all night as well as all day, being attended by a hardy group of young hard-rock miners able and eager to put in a 72-hour shift. For Tuesday evening, two annual dinners were scheduled, one for the ladies, which included a fashion show by Eaton's, and another for the gentlemen. At the latter, the customary impressive reading of the citations for bravery was again in the most capable hands of E. A. Collins, who founded these awards. They are given every year to three miners who, at the risk of their own lives, have rescued their fellow workers from certain death. Also awarded were two student prizes, the John T. Ryan Safety Trophies (to mining companies), the Leonard Medal to P. T. Bloomer, the Barlow Memorial Medal to A. B. Yates of the International Nickel Co., and the Inco Medal to James Roycroft Gordon of the same company.

Dancing was confined to the last night of the meeting, the party beginning at 10 o'clock and still going strong at 1:45. No late meetings or entertainment are scheduled for the first two days of the meeting so that members can get to bed early and be ready for the technical sessions the next morning. At least that's what we were told.

Next year's annual meeting will be in April at Toronto. F. V. Seibert, CNR industrial commissioner at Winnipeg, succeeds Mr. Diamond as president. E. J. Carlyle, after eighteen years as secretary and treasurer of the CIM, is about to retire.

Surface plant of the Mayflower mine of the New Park Mining Co. near Keetley, Utah, consists of mine office, change house, shops, storage facilities, and loading trestle. Ore is loaded into trucks or railroad cars for transportation to the Midvale mill of the U. S. Smelting Refining and Mining Co.

Mining Is Fun at New Park

By JOHN V. BEALL • MANAGING EDITOR

When a mine has ore averaging 5% lead, 7% zinc, 0.60% copper, ¼ oz gold, and 6 oz of silver, adequate reserves, power and water, easy access to market, and is situated in beautiful natural surroundings with the advantages of a big city within a few hours drive, there is little left to be desired. Such a mine is the Mayflower mine of the New Park Mining Co. situated 30 miles from Salt Lake City in the Wasatch Mountains near Keetley, Utah. With these natural attributes, the little left to be desired is provided by the progressive management under the leadership of W. H. H. Cranmer, president and general manager, and Clark L. Wilson, superintendent.

Miners are organized in an independent union which is on the best of terms with the management. As much

Cut-and-fill stoping is used to mine ore veins which occur in fissures in a quartz diorite porphyry host rock. Here, a sublevel is advanced at the same time holes are drilled for the first cut.

On the right: loading holes preparatory for a blast. A floor has been laid to keep the ore blasted from mingling with the waste fill. Ore is slushed to the raise chute. Three compartment raises are driven on 60 to 100-ft centers.

XUM

Loading a train on the main haulage level from an air-operated chute. Chute pullers and trammers receive bonus for sorting.

of the work as possible is contracted. Stopes are measured bimonthly by the engineers and work paid for on the volume of rock excavated; development is paid on the linear foot of advance; special timbering is done for a predetermined price arranged by the superintendent. An interesting feature of the incentive payment program is the bonus for both ore and waste sorting to trammers. When pulling the 800-level ore pockets, the chute puller and trammer sort waste into an empty car and are paid by the carload. When pulling waste they sort ore in the same manner. All contracts are approved by the superintendent, who is always available to discuss disagreements. It is not unusual for good miners to make over $15 a day.

Employes have a real interest in the Company, due, in part, to progressive personnel practices. The Company is now experimenting with a "Share of Production Incentive Plan." Each month the employes receive a percentage of "production values" which is the difference between the operating cost and the income from ore sales. The Company also sponsors 60% of the cost of a retirement plan, life insurance, and insurance for non-occupational accidents. The personnel department aids the employe in obtaining hospitalization benefits.

Pumping station showing one of three 500 gpm Pomona deep-well pumps. About 720,000 gal must be pumped a day.

Hoisting in balance with this 150 hp underground hoist brings ore from the various levels of the 1060-ft shaft to the Mayflower tunnel level.

As shipshape as when installed six years ago, the WN114 Sullivan compressor driven by a 600-hp, 2300-v synchronous motor develops 300 cfm.

An industrial hygienist, F. F. Netzeband, conducts periodic ventilation surveys, dust counts, and air analyses to improve conditions in individual working places. Plans are under way for the construction of a model village where employes can acquire and own modern homes.

The Company encourages employes to participate in civic enterprises, believing that the continuation of the American way of life depends upon

the support of each individual citizen. The basic policy of Company operation is to recognize the mutual rights and responsibilities of management, employes, stockholders, and the public. When this philosophy is practiced in every day relationships it is little wonder that mining is fun at New Park.

F. F. Netzeband, industrial hygienist, makes regular dust surveys to improve conditions in the working places.

Navy surplus blowers, bought at bargain prices, are used to bring air to the working face in stopes, drifts, and raises.

This ventilation raise is covered by permanent grill work to prevent accidents.

This miner uses safe practice in handling powder, opening boxes with wooden wedge and mallet. Powder room is coated with gunite to reduce moisture.

As part of the safety program, which is emphasized in all phases of work, training is given in the use of oxygen breathing apparatus and mine rescue methods.

This 5-ton loco is used in the Mayflower tunnel to bring ore to the surface. One ton battery motors are used on transfer levels. On the right: Shaft stations are kept clean. Chain over track keeps cars from rolling into the gate. Hoist drum for chippy hoist is set over shaft.

Some Variables Affecting Countercurrent Decantation

By E. A. PERETTI • ASSOCIATE PROFESSOR OF METALLURGY, UNIVERSITY OF NOTRE DAME; MEMBER, AIME

Since its development about forty years ago by J. V. N. Dorr (Cyanidation and Concentration of Gold and Silver Ores, McGraw Hill Book Co., 1936, 1-5), and others, continuous countercurrent decantation has not only become one of the standard methods for extracting gold and silver from their ores, but has also found wide application in other metallurgical and chemical industries.

In 1917 L. B. Eames published an excellent paper (Countercurrent Decantation, Trans AIME, 1917, *57*, 143-158), in which he determined the effects of grade of ore, ratio of solution precipitated to ore treated, variations in the density of the thickener spigot discharge and variations in the value of the barren solution upon the gold assay of the solutions discharged from the thickeners in the system. Eames showed that, other things being equal, the dissolved gold lost in the tailing varies directly as the value of the ore and as the value of the barren solution. The tailing loss decreases rapidly as the ratio of solution precipitated to ore treated is increased to about three or four to one. Beyond this point the improvement is negligible.

It is the object of this paper to present results of calculations for cyanidation in which four additional variables are considered, namely: the point of addition of barren solution to the circuit, the number of secondary thickeners, the percent of the soluble gold dissolved in the grinding circuit, and the omission of the primary thickener. Some of the variables considered by Eames and some of those to be discussed are subject to control while others are not.

The method for solving problems of this nature is well discussed elsewhere (J. V. N. Dorr: loc. cit. 133-139. A. Butts: Metallurgical Problems, 2 ed., McGraw Hill Book Co. 1943. 284-287) and hence won't be repeated here. There will be considered a common type of flowsheet, Fig. 1, consisting of one primary and four secondary thickeners having a spigot product with 50 percent solids. It is assumed that 75 percent of the values are dissolved in the milling circuit and 25 percent in the agitators and that the liquid-to-solids ratio of the feed to the primary thickener is five to one. Dissolution in the secondary thickeners will be neglected. Furthermore, as is customary, it will be as-

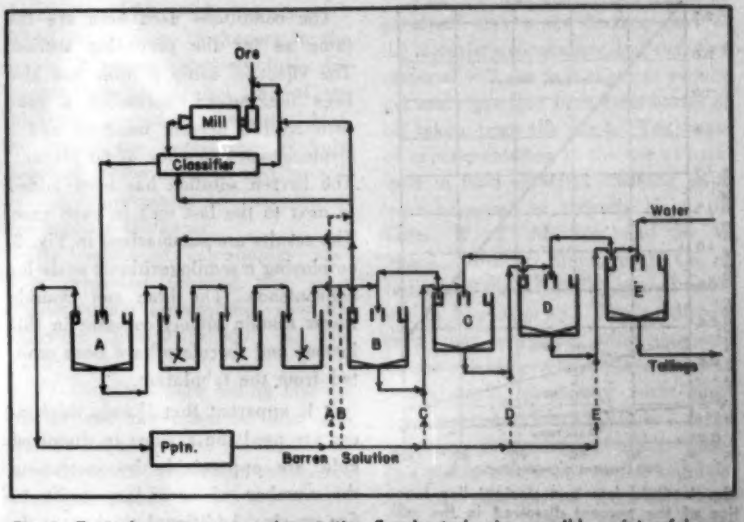


Fig. 1. Typical countercurrent decantation flowsheet showing possible points of barren solution addition.

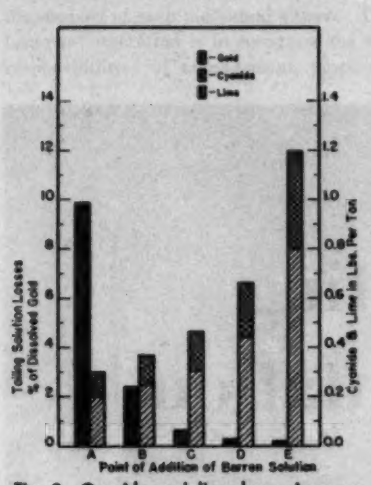


Fig. 2. Cyanide and lime losses increase and gold decreases the farther toward the end of system barren solution is added.

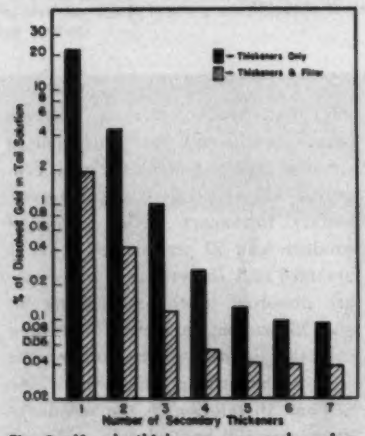


Fig. 3. If only thickeners are used, saving in gold is appreciable if the number of secondary units is increased to five or six.

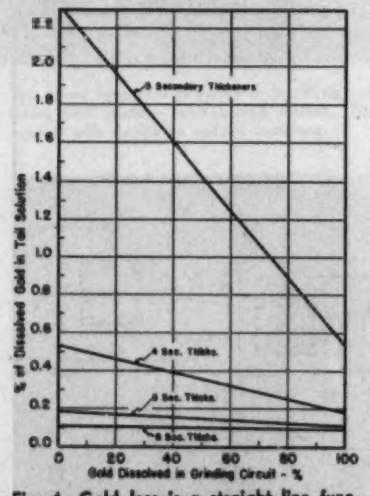

5 Secondary Thickeners
4 Sec. Thkns.
5 Sec. Thkns.
6 Sec. Thkns.
% of Dissolved Gold in Tail Solution
Gold Dissolved in Grinding Circuit - %

Fig. 4. Gold loss is a straight line function of the percent dissolved in the mill and the more thickeners, the less the effect of variation in point of gold dissolution.

sumed that the cyanide and lime strengths of the solution leaving the primary thickener will remain constant. For calculation we shall use 1 lb of cyanide, a protective alkalinity of a ½ lb of lime per ton of solution, and a barren solution containing gold to the extent of 2 cents per ton.

Upon making the foregoing assumptions and computing the assays of the tail solutions in gold, cyanide, and lime with the addition of the barren solution to thickeners A, B, C, D, and E, respectively (see Fig. 1), the tailing solution losses shown in Fig. 2 were obtained. These clearly show that the dissolved cyanide and lime losses increase and the dissolved gold loss decreases the farther toward the end of the system the barren solution is placed. Also, it can be seen that the loss in cyanide and lime is almost halved by placing the barren solution into thickener "D" rather than "E," while the gold loss is not markedly increased. By introducing the barren solution into "C" rather than "D," as great a saving in lime and cyanide is not made and the gold loss almost triples. This is perhaps the main reason that in actual plants one most often sees the barren solution being added in the next to the last unit in the series. Other things being equal, the most economical place to add the barren solution to the system will be determined by the unit value of gold, lime, and cyanide.

The conditions used here are the same as for the preceding section. The effect of using a filter has also been determined, assuming a cake with 33 1/3 percent moisture and a displacement efficiency of 60 percent. The barren solution has been added to next to the last unit in each case. The results are summarized in Fig. 3, employing a semilogarithmic scale for convenience. The lime and cyanide losses remain almost constant in this instant and therefore have been omitted from the tabulation.

It is apparent that if only thickeners are used the savings in dissolved gold are appreciable by increasing the number of secondary units to five or six. Additional thickeners do not bring corresponding improvements. As is well known, the use of a filter decreases the dissolved losses still further; but again, little improvement is achieved by using more than four or five thickeners and a filter. From the economic viewpoint the proper number will be dictated by the cost of thickeners and filters when balanced against the potential losses.

Calculations of the dissolved gold loss were made, using previously stated assumptions and varying the gold dissolution in the milling circuit. The number of secondary thickeners used varied from two to seven. The results are shown in Fig. 4. As is evident from Fig. 4, the dissolved gold loss is a straight line function of the percent dissolved in the mill. (In order not to confuse the diagram the graphs for the two and seven thickener cases have been omitted.) It is also obvious that the greater the number of secondary thickeners used, the lesser will be the effect of variation in point of gold dissolution upon the dissolved loss.

Fig. 5 is the result of plotting the logarithm of the slope of the straight lines obtained in Fig. 4 versus the number of secondary thickeners used. By combining Fig. 4 and 5 and letting

X = percent of gold dissolved in the mill

Y = percent of dissolved gold lost in tail solution

Z = the number of secondary thickeners

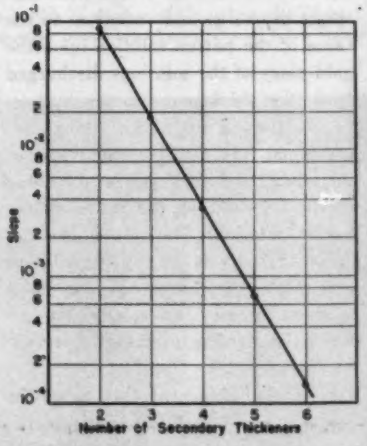


Fig. 5. The slope of the lines of Fig. 4 versus the number of secondary thickeners.

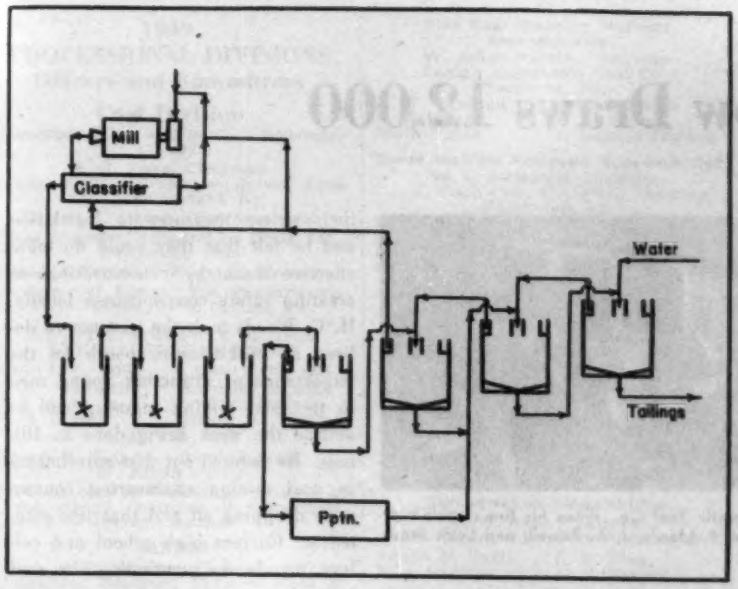


Fig. 6. Calculations indicate that lower dissolved gold losses are obtained if no primary thickening is done as shown in this flowsheet.

we obtained the following relationship:

$$X + 10^{(0.7Z-0.549)}\ Y = 125 + 0.089 \times 10^{(0.7Z-0.549)}$$

This equation applies, of course, only to the cases where the conditions are those assumed in the paper. Other conditions will give a similar identity, but with different constants.

Perhaps most of the time in the application of countercurrent decantation systems a primary thickener is essential in order to obtain a pulp of suitable consistency for agitation. Calculations indicate, however, that lower dissolved gold losses are obtained if no primary thickening is done, with a setup as shown in Fig. 6. It is clear that with this arrangement it will make no difference how the dissolution is distributed between the grinding circuit and the agitators. The losses are given in Fig. 7 in comparison with those occurring when the primary thickener is used assuming 10 percent of the gold dissolved in the grinding circuit. The saving is great if up to about five thickeners are used. With six or more thickeners in the system it makes relatively little difference whether or not one of the thickeners is used as a primary. Also, the larger the percentage of gold dissolved in the mill, the smaller will be the improvement by omission of the primary thickener.

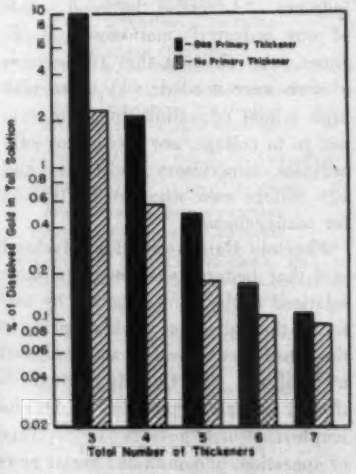


Fig. 7. Comparison of dissolved gold losses with and without primary thickening; up to about 5 thickeners, the saving is great.

The results of calculations have been presented showing that in a countercurrent decantation system the following generalizations can be made:

(1) Dissolved gold tailing loss decreases and the dissolved lime and cyanide losses increase the farther toward the end of the system the barren solution is placed.

(2) Dissolved gold loss decreases rapidly as the number of thickeners is increased to five or six. Beyond this number, the loss decreases very little.

(3) Dissolved gold loss varies inversely as the percent dissolved in the mill.

(4) If less than five thickeners are used, the dissolved gold loss will be less by not using one of them as a primary, if it is possible to do so.

Weeds, Wood, and 2-4-D

"Come forth into the light of things,
Let Nature be your teacher."
—Wordsworth

Mine operators who are having trouble with fungus-infested timbers are urged, by one of their Vermont confrères, to spend a little time on their front lawns thinking it over.

Willis P. Mould of Williamstown notes in a recent letter that he had been warring against weeds with a preparation called 2-4-D, and, entering his cellar, bespied fungus growth upon his coal bin. "I gave it a shot of that 2-4-D," he says, "and within a week it disappeared"—never to return again. Mr. Mould is cautious, lest magical powers be ascribed to 2-4-D, but he believes it's worth a fair trial. If unsuccessful on the timber, it's always useful on the front lawn.

More Gas from Each Barrel

"Fill 'er up with silica-magnesia catalyst" may never become part of the motorist's vocabulary, but that new material will see to it that car owners get more gasoline from each barrel of oil taken from the earth. Ten years of experimentation in the use of catalysts in fluid catalytic cracking units were reported on recently by A. L. Conn, W. F. Meehan, and R. V. Shankland of the Standard Oil Co. of Indiana at the AIChE's regional meeting in Los Angeles. They've discovered that synthetic silica-magnesia causes cracking to take place at lower temperatures, producing more gasoline, less coke. Tests made in a large cracking plant showed higher gasoline yields with lower octane number, but the addition of tetraethyl lead overcame the latter difficulty, their report said.

AMC Coal Show Draws 12,000

W. G. Adams (left), of the Chicago Pneumatic Tool Co., shows his firm's exhibit at the AMC coal convention to L. E. Young, W. P. Morris, J. A. Russell, and Louis Frost.

While applying the first coat of paint to some new screens the morning after the 1949 coal show of the American Mining Congress, held in Cleveland, May 9 to 12, it struck me that there was a striking parallel between this somnolent process and attendance at an exhibition of that size. No matter how carefully you apply that first coat, there are always some bare spots. So it is with surveying an exhibition of over 200 manufacturers in which all types of coal mining and cleaning equipment are displayed; you can't see everything. Another source of bare spots at that convention might arise from a different sort of painting (key words: *red* and *town*) or from persistent attendance at the numerous cocktail parties and open houses. While we're on the subject of entertainment, the Coal Miners' Party was a gay gathering at which guests partook of buffet supper, free beer, entertainment, and dancing, while the Annual Banquet starred Frances Langford.

After chatting with a gentleman from England, snapping a picture of two from Canada, and seeing a representative of the coal mining industry from India, I can safely say that the 12,000 people registered at the show came from all over the world. The Cleveland Public Auditorium is probably one of the few buildings in the world that could contain the magnificent equipment display and the throng that came to learn and admire. Boutonnières were pinned on spectators; gifts such as canes, buttons, notebooks, key holders, and pencils were profusely distributed; and everywhere people were loaded down with manufacturers' literature. All types of coal mining and cleaning equipment such as cutting machines, loaders, shuttle cars, locomotives, draglines, bulldozers, excavators, trucks, safety appliances, conveyors, screens, heavy-density plants, rock drills, and electrical equipment, to name a few, were displayed.

Meeting programs were not scheduled for Monday morning or Thursday afternoon so that exhibits could receive undivided attention, but the intervening days were replete with motion pictures, many through the courtesy of the Bureau of Mines, and technical sessions.

Probably the most interesting group of talks was given at the management session on Thursday morning which was ably chairmaned by L. C. Campbell. On the subject of management-employe responsibility for mine safety, Charles W. Connor outlined three good points for reducing accidents: (1) pre-job training for new men, (2) safety training for supervisors by the Bureau of Mines, (3) safety training of employes by management. Mr. Connor said that unions had confined their safety measures to legislation and he felt that they could do more effective work by concentrating on creating safety consciousness locally. H. C. Woods took the rostrum to deliver an enthusiastic speech on the importance of attracting young men to the coal mining industry and to outline the work being done in Illinois. He pointed out that enrollments in coal mining engineering courses were dropping off and that few studied it. Current high school and college text books mentioning the coal industry carry information on the 1895 to 1910 period, referring to the industry as unhealthy and dangerous. Mr. Woods recommended that we contact our local high schools and acquaint them with the coal industry.

The coal operators of Illinois, in co-operation with the University of Illinois, are sponsoring a one-week course about the coal industry for educators. Expenses for this course for high school and college teachers are paid by the operators. C. E. Lawall seconded Mr. Wood's talk by affirming that industry must take the initiative in making young men feel at home and point out the opportunities present in a career in the coal industry. "Advertise the good points of our industry" maintained D. C. Jones, who claimed that two classes of men were needed: (1) those with high school education, but who cannot go to college, are needed for subordinate supervisory positions, and (2) college men who can be trained for management.

Ebersole Gaines and Live Treland said that better management-employe relations could be maintained by fostering the individual worker's pride in the job. This can best be accomplished by publicity. Sam Cassidy pointed out that a company publication for its employes which gave broad coverage of operation, personal, and social news could do much to improve personnel relations.—J.V.B.

1949
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Further Consideration Given by Board to Report of Democratization Committee

Chief topics for discussion at the meeting of the Board of Directors, AIME, in New York on Wednesday, April 20, were the report of the Committee on Democratization and the forthcoming referendum on increased dues. Eighteen Directors, including two alternates, spent 5½ hours in their deliberations, which started with a luncheon meeting at the Engineers' Club, and continued in the Board room at Institute headquarters. President Young presided, with the following directors in attendance: Messrs. Bowles, Daveler, Elkins, Head, Kinzel, Kraft, Meyerhoff, Millikan, Peirce, Schumacher, Suman, Swift, Weed, Wrather, and Fletcher. Mr. Lovejoy acted as alternate for Mr. Watson and Mr. Work substituted for Mr. King.

The essential points made by the Committee on Democratization were reported in the March and April issues of the AIME journals. Mr. Gillson reviewed the report further, emphasizing that the Committee would be satisfied if the spirit of its report were accepted; for the time being, at least, it realized that certain of its suggestions, such as greatly increased traveling by the Secretary, a field secretary for the Mining Branch, and paid expenses of two Local Section delegates to three meetings a year, would be more costly than the Institute would be justified in undertaking. The Board was especially interested in the plan for a more formal organization of the Section delegates' group, and asked the Committee to work up a specific plan for presentation at the September meeting of the Board. There was some discussion as to the desirability of having fewer meetings of the Board, but making them more important, with carefully prepared agenda; it was thought that this would not be opposed to the idea of democratization. The Petroleum Branch reported its feeling that extensive trips into the field by the Secretary should not be necessary if he kept in close touch with the Branch executive committees, who would in turn, keep themselves fully informed of the feeling of the membership at large. The Branch also favored an increased number of younger men on the Board.

As to the progress of the movement to increase dues, approval was voted of the statement appearing over President Young's signature in the May issues of the journals. This was supported by separate editorial appeals in the three journals and mimeographed data prepared by the Secretary's office and distributed to all Directors and Local Section Chairmen. Considerable discussion arose as to the form of ballot for the referendum, and it was finally decided that this should take the form of a postal card which would be signed by the voter, who would not only indicate his approval or disapproval of the proposed increase, but would indicate his Branch affiliation. A reprint of President Young's statement would accompany the ballot. The postal card would be stamped and self-addressed to the Institute, and would be mailed on or about May 20 to all paid-up AIME members in the United States, Can-

AIME STAFF

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Assistant Treasurer H. A. MALONEY
Assistant Secretaries E. J. KENNEDY, JR.
ERNEST KIRKENDALL
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(Dallas, Texas)
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ada, and Mexico, returnable by June 10, so that the results can be reported by officially appointed tellers, representing each Branch, at the June 15 meeting of the Board. The form of referendum ballot is not specifically stated in the bylaws but counsel advised that the proposed postal card meets the requirements for a "letter ballot."

deficit to decrease in 1949

Based on operations for the first quarter, the final budget for 1949, as offered by Mr. Daveler, shows income at $380,350 compared with total expenditures of $437,479, or a deficit of $57,129, to be in part offset by voluntary contributions of $8,500, or a net estimated deficit for the year of $48,629. This would compare with the deficit of $82,000 suffered in 1948. The reduction in the deficit is almost exactly counterbalanced by the expected increase in advertising revenue in 1949 from the three new journals. Details of the budget, which this year for the first time is apportioned to the three Branches, were offered for discussion by the Board. The paid membership is divided approximately in the ratio of 52 percent for the Mining Branch, and 24 percent each for the Metals and Petroleum Branches. Allocating income and expenses to Branches, on the basis of information supplied by members, the deficit for the Metals Branch in 1949 is estimated at $55,122 and for the Petroleum Branch $17,161, whereas the Mining Branch would show a surplus of $23,754.

In view of the fact that many 1948 publications are out of print, the Board voted to allow a credit of $5 to all members in the United States who pay up their dues for 1948 and who cannot be supplied with one or more requested publications. In view of the further difficulty that some foreign members have had in securing American dollars, it was further voted to remit the entire amount of 1948 dues to those foreign members now in arrears for 1948 and 1949 dues. Also, the time was extended to Dec. 31, 1949, in which Junior Members of ten years' standing may change their status to Member or Associate Member without payment of an initiation fee; the date had previously been set at July 1, 1949.

Approval was voted to recommend to the Engineering Foundation that grants be made for certain research projects as follows: $2,500 for Project 97, Diffusion in Steel, by Morris Cohen; $3,000 for Project 98, Mechanical Properties of Binary Alloyed Ferrites at Low Temperatures, by R. M. Brick; and $1,000 for a new project in the field of comminution, by A. Kenneth Schellinger.

1951 annual meeting in St. Louis

Formal acceptance was voted of the invitation of the St. Louis Section to hold the 1951 Annual Meeting of the AIME in St. Louis, which is probably the most centrally situated city in which an Annual Meeting can be held for the convenience of AIME members. The 1950 Annual Meeting will be held at the Hotel Statler in New York, Feb. 12-16. No decision has yet been made as to the scene of the Fall Meeting of the AIME in 1950; it will be impractical to hold it in New Orleans as suggested.

Erle V. Daveler was appointed to represent the Institute on the John Fritz Medal Board of Award succeeding Clyde Williams, resigned. Fred Searls, Jr., was named to succeed H. M. Lavender as chairman of the Saunders Mining Medal Committee, and H. L. Roscoe was named to take Mr. Lavender's place as a committee member. Oliver C. Ralston was reappointed as an AIME representative on the National Research Council. Albert M. Portevin was named to represent the AIME at the centenary celebration of the French Society of Metallurgy in Paris next October.

The Cornell Metallurgy Society, at Cornell University, was formally recognized as an Affiliated Student Society, AIME. Selection of James A. Stangarone as advertising manager of the *Journal of Metals* was reported.

Print Orders for Journals

Institute headquarters was faced with an important problem last December when it became necessary to decide on how many copies of each of the three journals to print, with no knowledge of what the wishes of the members would be. We wanted stocks to be adequate to supply the demand, but did not wish to waste money in printing journals for which there was no need. The print orders for the January issues were finally set at 11,500 for *Mining Engineering*, 6700 for the *Journal of Metals*, and 5500 for the *Journal of Petroleum Technology*. In the light of subsequent information only slight changes had to be made in succeeding issues: 10,800, 6700, and 5400 respectively. All issues of each magazine could be supplied up to May 1 except the January issue of the *Journal of Petroleum Technology*.

Progress of Publication of AIME Volumes

In the December issue of *Mining and Metallurgy*, page 686, was published a list of the volumes whose publication was expected in 1949 and the terms under which members could secure them. The expected dates of publication of these volumes are:

No. 175 Institute of Metals Division. May.

No. 176 Iron and Steel Division. August.

No. 177 Coal Division. August.

No. 178 Mining Geology. May.

No. 179 Petroleum Development and Technology. August.

No. 180 Institute of Metals Division. October.

No. 181 Metal Mining, etc. October.

No. 182 Nonferrous Smelting and Refining. October.

No. 183 Minerals Beneficiation. November.

Open Hearth Proceedings. November.

Blast Furnace Proceedings. November.

Electric Furnace Proceedings. September.

Rod and Wire Symposium. August.

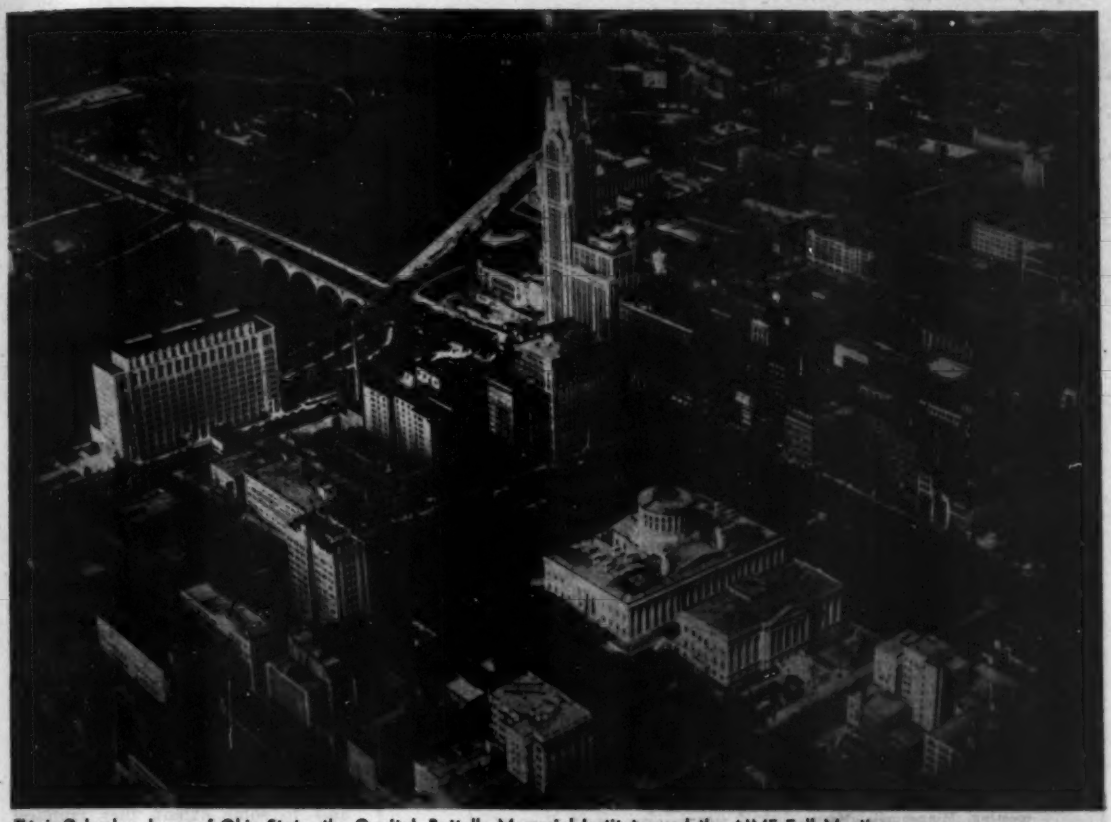
Statistics of Oil and Gas. August.

Industrial Minerals and Rocks. 2d ed. August.

Berwerk—und Probierbuchlein. Now available. $3 members; $5 nonmembers.

Conservation of Petroleum. No date set.

Coal Preparation. Revised edition authorized but no date set.



This is Columbus, home of Ohio State, the Capital, Battelle Memorial Institute, and the AIME Fall Meeting.

AIME General Meeting Set for September

The date has been set, and tentative plans prepared, for the AIME Fall Meeting, to be held this year in Columbus, Ohio. The kickoff, and it will be a real kickoff, is set for Saturday, Sept. 24, which day will be spent at the Ohio State–Missouri football game in Columbus. Registration will be the following day, and Monday will see the beginning of a six-day round of technical sessions, inspection trips, and executive conferences. The meeting will close on Oct. 1, just in time for the Ohio State–Indiana football game.

Twenty-one hours of technical sessions will be held on Monday, Tuesday, Wednesday, and Friday, but Thursday will be reserved for inspection trips which will include the National Electric Coil Co., Jeffrey Mfg. Co., and the Battelle Memorial Institute. Programs are being planned by most of the Divisions of the Institute. A cocktail party and banquet will also be held during this busy week. In addition, the Southern Ohio Section of the Open Hearth Committee, AIME, will meet on Friday, Sept. 30, for technical sessions, luncheon, and an evening banquet. The Open Hearth group's Executive Committee will also meet on the last day of the Meeting, and then be off to see the closing grid-iron tussle.

The Neil House in Columbus will be headquarters for this mid-year general meeting, and those who are planning to attend can write Hugo E. Johnson at the Battelle Institute for further details.

AIME Research Grants Continue

Four AIME research projects, dealing with diffusion of steel; alloys of iron research; basic open hearth steelmaking; and the behavior of metals during hot-working, will continue to receive support this year from the Engineering Foundation, the research department of United Engineering Trustees, Inc., for the Founder Societies, a group founded thirty-five years ago for the ASCE, ASME, AIEE, and AIME.

Although the Foundation's available funds are now only $24,353 or less than half what they were a year ago, that amount will provide grants for a total of thirteen research projects this year. Contributions are made primarily to support studies and analyses necessary to organize projects to a point where they can be supported from other sources. Eighteen such projects were supported during the past year.

EJC Survey on Engineering Employment and Salaries

The General Survey Committee of the Engineers Joint Council, which is composed of representatives of the major engineering societies, has recently concluded a survey of representative industrial companies and governmental agencies concerning their 1949 employment programs for engineering graduates. Of 500 questionnaires sent out late in January, 193 had been returned early in May, but none of these were from strictly mining companies.

Excluding a few small organizations providing incomplete data, there were 162 industrial companies and 31 governmental agencies in this group with over 4 million employes, of whom 89,000 were engineering graduates, or a third of some 265,000 such graduates in the country.

These organizations reported that they had hired 10,390 inexperienced engineering graduates in 1948 and estimated that they would hire 8174, or some 21% less in 1949. For the industrial companies, the decrease was 26% to a total of 5695; for the governmental agencies, 9% to one of 2479. Agencies of the state governments actually expected a substantial increase.

On the basis of the starting rates reported, 25% of these men at the Bachelors' level would receive a starting rate of $275 per month or more, 50% one of $255 or more (the so-called median rate) and 75% one of $250 or more.

The median rates reported by organizations, without regard to the numbers of men to be employed, were $255 for Bachelors', $300 for Masters', and $375 for Doctors' degrees.

A median monthly increase of $30 is given by the end of the first year by the organizations in this group. 45% of them give this total at one time, 49% in two installments, and the remainder in three or four installments.

In order to secure some indication of the recent trend in salaries of engineering graduates with experience, the survey also requested data for those 10 years beyond their Bachelors' degrees late in 1946 and those who were 10 years out late in 1948. Only 82 organizations provided these data and, to some extent, they represent estimates rather than exact figures. For 1511 graduates 10 years out in 1946, the median monthly salary was $375 and for 1431 who were 10 years out in 1948 it was $450, or 20% more. The 1946 figure corresponds closely with that secured by a survey of individual engineering society members at that time, and on that basis the current data for both years may be regarded as reasonably reliable.

Calendar of Coming Meetings

JUNE

2 Reno Branch, Nevada Section, AIME.
3 Columbia Section, AIME.
3-4 Central Appalachian Section, AIME, spring meeting, Blacksburg, Va.
5-10 SAE, summer meeting, French Lick, Ind.
8 El Paso Metals Section, AIME.
13-15 15th National Applied Mechanics Conference, University of Michigan, Ann Arbor.
13-15 Annual convention, Idaho Mining Association, Sun Valley.
14 East Texas Section, AIME.
15 Southwest Texas Section, AIME.
16 Carlsbad Potash Section, AIME.
16 Engineering Foundation, 29 W. 39 St., New York City.
June 27-July 1 ASTM, 52nd annual meeting, Chalfonte-Haddon Hall, Atlantic City, N. J.
20-22 American Society of Heating and Ventilating Engineers, semi-annual meeting, Minneapolis, Minn.
20-24 AIEE, summer general meeting, New Ocean House, Swampscott, Mass.
20-24 American Society for Engineering Education, RPI, Troy, N. Y.
21 Gulf Coast Section, AIME.
25-26 American Iron and Steel Institute, general meeting, N. Y. C.
27-30 ASME, semi-annual meeting, San Francisco.
27 Alaska Section, AIME.
28 Montana Section, AIME.
29 Pennsylvania - Anthracite Section, AIME, summer meeting Split Rock Lodge, Pocono Mountains.

JULY

9-23 Fourth Empire Mining and Metallurgical Congress, Great Britain
13-15 American Society of Civil Engineers, summer convention, Mexico City.
15-24 Pan-American Engineering Congress, Rio de Janeiro.

AUGUST

23-26 AIEE Pacific general meeting, Fairmont Hotel, San Francisco.
26-Sept. 10 Engineering & Mining Exhibition, London, England.

SEPTEMBER

25-28 Mid-year Meeting, AIME, Neil House, Columbus, Ohio.
25-28 American Mining Congress, Western fall meeting, Spokane.
29-30 ASME, fall meeting, Erie, Pa.
29-Oct. 1 Colorado School of Mines, 75th anniversary celebration.
Sept. 30-Oct. 1 Southern Ohio Section of Open Hearth Committee, AIME, fall meeting, Deshler-Wallick Hotel, Columbus.

OCTOBER

5-7 Petroleum Branch, AIME, fall meeting, Plaza Hotel, San Antonio, Texas.
13-14 Texas Mid-Continent Oil and Gas Association, annual meeting, Rice Hotel, Houston.
14 Eastern Section, Open Hearth Committee, Iron and Steel Division, annual all-day fall meeting, Warwick Hotel, Philadelphia.
14 Southwestern Section, Open Hearth Committee, Iron and Steel Division, Kansas City, Mo.
17-19 Institute of Metals Division, AIME, fall meeting, Allerton Hotel, Cleveland.
17-23 AIEE, 1949 Mid-West meeting, Netherland Plaza, Cincinnati.
20-21 Petroleum Branch, AIME, Elks Club, Los Angeles.
24-28 Thirty-seventh National Safety Congress and Exposition, Chicago.
26-27 Joint Fuels Conference, ASME-AIME, French Lick Springs Hotel, French Lick, Ind.
28 Pittsburgh Section of Open Hearth Committee and Pittsburgh Section, AIME, annual fall meeting, William Penn Hotel, Pittsburgh.
28-29 ECPD, annual meeting, Edgewater Beach Hotel, Chicago.

NOVEMBER

1-5 Pacific Chemical Exposition, California Section, American Chemical Society, San Francisco Civic Auditorium.
2-4 American Society of Civil Engineers, fall meeting, Washington, D. C.
7-10 AIChE, annual meeting, Pittsburgh, Pa.
9-11 Industrial Minerals Division, AIME, Tampa, Fla.
12-14 Geological Society of America, annual meeting, Hotel Cortez, El Paso.

DECEMBER

8-10 Seventh Annual Conference, Electric Furnace Steel Committee, Iron and Steel Division, AIME, Hotel William Penn, Pittsburgh.

JANUARY 1950

18-20 American Society of Civil Engineers, annual meeting, New York.
30 AIEE, winter meeting, New York.

FEBRUARY 1950

12-16 Annual Meeting, AIME, Statler (Pennsylvania) Hotel, New York City.

APRIL 1950

10-12 Open Hearth Conference, and Blast Furnace, Coke Oven and Raw Materials Conference, Netherland Plaza Hotel, Cincinnati.
19-21 American Society of Civil Engineers, spring meeting, Los Angeles.

NOVEMBER 1950

16-18 Geological Society of America, annual meeting, Hotel Statler, Washington, D. C.

DECEMBER 1950

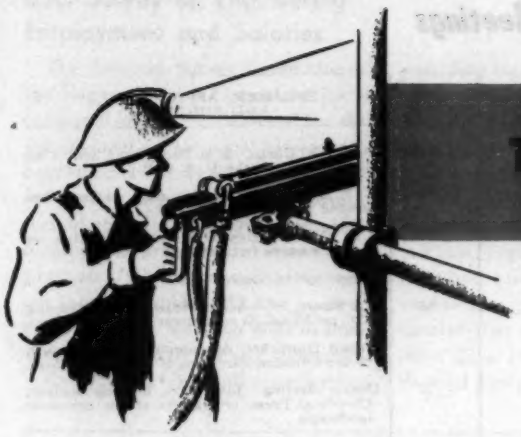
7-9 Eighth Annual Conference, Electric Furnace Steel Committee, Iron and Steel Division, AIME, Hotel William Penn, Pittsburgh.

If You Are Visiting Dayton

AIME members visiting Dayton, Ohio, are cordially invited to make use of the facilities of the Engineers' Club of Dayton, 112 E. Monument Ave., one block from the downtown business district. A guest card good for two weeks will be provided upon proper identification. Sleeping facilities are not available, but regular luncheons are served, a reading and billiard room is provided, and there is an excellent library of 10,000 volumes, with 100 periodicals received. William Siler is manager.

What Went on at Recent Local Section Meetings

SECTION	DATE	PLACE	PRESIDING OFFICER	ATTEND-ANCE	SPEAKER, AFFILIATION, AND SUBJECT
Boston	April 4	MIT	George P. Swift	146	Clifford L. Curan, E. B. Badger and Son" o'An Engineer Views the U. S. S. R."
Carlsbad Potash	Mar. 24	Carlsbad	J. P. Smith	69	C. P. Theis, New Mexico state geologist, on ground water.
Carlsbad Potash	April 21	Carlsbad	J. P. Smith	57	E. F. Bunning, Central States Cooperative, on potash's use in fertilizers.
Carlsbad Potash	April 23	Carlsbad	J. P. Smith	146	Annual spring dance.
Chicago	April 6	Chicago Bar Association	T. S. Washburn	83	J. S. Smart, Jr., American Smelting and Refining Co., on ASARCO continuous casting process.
Cleveland		Cleveland			Harold Hunsicker, Aluminum Co. of America, on precipitation hardening of aluminum alloys.
Cleveland	April 14	University Club		22	Dinner meeting. John Love, business analyst, Cleveland Press, on aspects of the economic landscape.
Columbia	April 1	Spokane, Idaho	Howard P. Sherman	61	Robert D. Dellwo, attorney, on Communist technique and strategy.
Connecticut	April 13	Bridgeport	H. L. Burghoff	58	A. D. Eplett, Manning, Maxwell and Moore, "Technological Observations in Postwar Germany."
Delta, Mississippi Subsection	April 21	Laurel, Miss.	R. E. France	82	Walter F. Rogus, Gulf Oil Corp., on condensate well corrosion. E. N. Dunlap, California Research Corp., on AIME Annual Meeting.
Detroit	April 18	Detroit	Eric Weinman		W. L. Grube, General Motors Corp., on the electron microscope as a metallurgical tool.
El Paso Metals	April 13	Texas College of Mines	Jack Despins	38	Baxter Polk, Texas College of Mines and Metallurgy, on the library for student and professional engineers.
Lehigh Valley	Mar. 25	Bethlehem	F. E. Van Voris	103	Bruce S. Old, Arthur D. Little, Inc., on radioactive tracers in metallurgy and coke production. A. M. Gaudin, MIT, on radioactive tracers in mineral engineering.
Montana	Mar. 29	School of Mines, Butte	Hollis McCrea	130	S. P. Nile, Montana School of Mines, on radioactivity.
Montana	April 26	Anaconda Copper M. Corp.			Visit to Anaconda ferromanganese plant. R. H. Townsend, superintendent of zinc roasters, on ferromanganese operations.
New York	Mar. 10	Mining Club	Thomas G. Moore	69	Claude A. Williams, President, Transcontinental Gas Pipe Line Corp., on natural gas.
New York	April 13	Mining Club	T. B. Counselman	87	J. A. Retty, chief geologist, Labrador Mining and Exploration Co., on the Hollinger-Hanna Iron Ore development in New Quebec and Labrador.
Nevada, Reno Branch	Feb. 3	Reno	L. D. Gordon		Silas E. Ross, Chairman, Board of Regents, University of Nevada, on The Board of Regents and the Law.
Nevada, Reno Branch	May 5	Reno	A. C. Johnson	44	Business meeting. Mackay School of Mines Day.
North Pacific	April 21	Seattle	Kenneth Anderson	52	Field trip, Boeing Airplane Co.
Pennsylvania-Anthracite	April 22	Altamont Hotel, Hazelton	Harry Otto		Spring meeting. S. H. Ash, chief, Safety Branch, U. S. Bureau of Mines, "Anthracite Flood-Prevention Project."
St. Louis	Jan. 21	Hotel York	Herbert A. Franke	70	Davis Read, chief engineer, West Kentucky Coal Co., on Coal Mine Mechanization. C. E. Davis, Goodyear Rubber Co., comments on film, "The Way Out."
St. Louis	Mar. 18	Hotel York	Herbert A. Franke	36	Fred B. Loomis, Shell Oil Co., on oil exploration. Paper prepared by Ernest G. Robinson.
St. Louis	April 22	Hotel York			J. G. Evans, Gardner-Denver Co., motion pictures taken in South Africa.
San Francisco	April 13	Engineers Club	E. H. Wisser	39	James T. Kemp, American Brass Co., on Cornish mine pumping engines.
San Francisco	Mar. 9	Engineers Club	E. H. Wisser	84	Edward Morris. Movies on California's gold mines.
Southwestern New Mexico	Mar. 16	Bayard	Joseph H. Taylor	27	W. H. Goodrich and W. R. Storms. Reports on AIME Annual Meeting.
Southwestern New Mexico	April 13	Bayard	Joseph H. Taylor	61	E. A. Slover, Chino Mines Division, Kennecott Copper Corp., on practices at Hurley copper smelter. E. M. Tittmann, American Smelting & Refining Co., "El Paso Zinc Fuming Plant."
Ohio Valley	April 11	Columbus	William Mueller	40	William Mueller, Department of Metallurgy Ohio State University, delegates report on annual meeting. Hugo Johnson, Battelle Memorial Institute, explanation of new Section by-laws and discussion of plans for mid-year or Regional meeting.



THE DRIFT OF THINGS

. . . as followed by EDWARD H. ROBIE

The Peak Is Passed

At last the postwar boom in metal prices has ended, and some mining companies, operating high-cost properties, will now have to curtail or close down. Copper at 23½¢ was high but not to be compared with lead at 21½¢, so the price of the former offered more resistance to decline, and began with but a slight shading by one seller. Lead, on the other hand took a headlong tumble, in less than a month, from 21½¢ to 16¢, and then dropped still further, to 14¢, by the middle of May. Zinc never got up to the price of lead, though before the war it ruled somewhat higher. From a high of 17½¢ it had dropped to 14¢ in mid-May. Copper was 18¢.

The high prices had the normal effect of stimulating an enormous amount of exploration and development work, and plant extensions. In a large number of instances the geologists have been successful, as Donald M. Davidson showed in his most interesting article in the March *Mining Engineering*. Important new metal mines are springing up all over the world. Unfortunately most of them will come into production when the demand has quieted down, and their added pressure on the market will drive prices still lower. Mineral production cannot respond to prices as quickly as can that of many other commodities.

Coal and oil, likewise, are in ample supply, but the price of oil never boomed like the prices of the nonferrous metals, so has better resisted a decline. Coal has been more nearly in line with the cost of production, so until wages come down, which we understand Mr. Lewis is not planning, no great cuts in price are likely there.

Though the metal mines will get less for their product from now on, there will be compensations. Labor will be easier to get and will work harder, and the cost of supplies will be less.

Good Times or Bad Times?

Our electrician, whom we called in the other day only to find a blown fuse that we did not have intelligence enough to find ourselves, told us that now he is really glad to have work, and no longer is trying to escape it. He says helpers are much easier to get too, and what's more they work instead of sitting around talking. This condition seems to be returning in all fields of activity. Even in professional mining, more people are looking for jobs, and less employers are calling for men. If not carried too far, the condition is a healthy one, and makes for better work habits. It will, of itself, reduce the number of jobs, however, for if everybody works ten per cent harder, then ten per cent less people will be needed to do the country's work. A diligent person, can, we suspect, still find something to do.

Buyers' Market for Graduates, Too

Engineering graduates this year are not able to sit back and take their choice of several offered jobs, which is the condition that has existed in recent years. The supply of graduates is increasing and the demand, owing to the recession in industrial activity, is decreasing. Many graduates that expected employers to come after them have found that they must go out and rustle jobs for themselves.

Employers tells us that where half the offers they made to students have been rejected in the last few years, the proportion is now only about 25 percent, and that most of this 25 percent is made up of men who plan to take graduate work or who have decided to work in local areas close to their homes. The increasing proportion of students that plan to take graduate work, in part because they cannot get attractive jobs, will make it just that much harder for those with Bachelors' degrees to get jobs in the years to come.

Also, we learn from employers that the character of the present crop of graduates has changed. Heretofore, most of them had had actual military service, and many had held commissions—in general a most desirable group, earnest, mature boys, trained in administration and supervision as well as in their technical specialty. Now, they are classifiable for the most part into two groups—(1) really outstanding boys and (2) graduates that should never have gotten that far in their schooling, the "4F's in the head," who went to college because there was nothing else to do and they could do so at the taxpayers' expense. They have not had much military experience—mostly just in camps. Even with a degree they will have rather hard sledding this year. The first group, however, the cream of the crop, should have no particular difficulty locating a satisfactory job even though they may have to go after it instead of it coming to them.

Gold Output Still Lags

U. S. Gold output last year was valued at 73 million dollars—just about half what it was before the war. The picture for the gold mines, however, is not as bright even as that might indicate, for most of the gold is obtained as a by-product, and would be produced whatever the price. Homestake is about the only straight gold mine of any importance left in the country. Canada has more gold mines by far; its output last year got back up to 123 million dollars, compared with 143 million in 1937. The freer labor supply and the current reductions in the price of supplies will redound to the advantage of the gold mines but it will take a real depression to make them prosperous. That is, unless the Government decides on what seems to us the sensible course of letting the gold miners sell their product freely for whatever it will bring. Certainly less than a hundred million dollars' worth of gold exported from the country in a year will not upset the international monetary commission.

Before we leave the subject of gold we might repeat an advertisement that the *New York Herald Tribune* reprints from an issue published 100 years ago: "Seekers after gold in the rivers and streams of California will need a suit of India-rubber to protect them from wet in their researches. Complete outfits of this material, coats, caps, boots, pantaloons with boots attached, camp blankets, life preservers, canteens, pails for collecting the gold &c. &c. for sale to enterprising adventurers at low prices by Ellis Tompkins & Co., New India-Rubber Warehouse, 28 John St." The pails for collecting the gold are now but rarely used.

$5-a-Ton Oxygen

Metallurgists have been using more oxygen in the last decade or so than ever before. Compared with air it speeds up reactions and thus gives added capacity. A considerable fillip to this use may result if its cost can be cut by any such figure as is indicated by news of a plant in Texas built by Dresser Industries. This plant is said to have attained an output of more than 175 tons of 90-95% oxygen a day at a cost of only $5 per ton. Although this is somewhat lower in grade than the 99% oxygen ordinarily used in large quantities, the cost is only about a tenth as much. The difference in grade should not be of any particular importance in most metallurgical processes, but any such difference in cost should open up an enormous new field of application.

We have not read the details of the new process other than that it depends on a low-pressure self-contained cycle, using only cooling water and natural gas for fuel. Air is compressed to 85 lb, and then cooled by expansion and self-refrigeration. Carbon dioxide and other impurities are removed by deposition on special aluminum fin-tube exchangers, and the oxygen and nitrogen separated in aluminum fractionating columns.

Scientific vs Legal Minerals

Down in Texas, Mrs. Emilie Heinat had willed surface rights to her 400-acre farm to a daughter, and the mineral rights to trustees for all of her children. When it was found that limestone could be produced commercially on the property, those to whom the mineral rights had been willed sued for possession of the property. The trial court held that limestone is a mineral, so the mineral owners should have the farm. The Court of Civil Appeals disagreed, and the Texas Supreme Court upheld the appeal. The interpretation of mineral rights was given by Associate Justice Smedley as follows:

"Substances such as sand, gravel, and limestone are not minerals within the ordinary and natural meaning of the word unless they are rare and exceptional in character, or possess a peculiar property giving them special value, as for example sand that is valuable for making glass, and limestone of such quality that it may be profitably manufactured into cement. Such substances, when they are useful only for building and road-making purposes, are not regarded as minerals in the generally accepted meaning of the word."

So, according to the newspaper report, "Limestone is not a legal mineral, although it may be a scientific mineral"! This is just so that our readers will be informed of the distinction, should they ever be asked.

An Apposite Title

Three years ago we suggested that the title "Honorable" might better be applied to engineers than to lawyers and politicians. There was no apparent stampede to adopt our excellent suggestion, but we note with approval that the seed we sowed at that time has finally sprouted. In its announcement of "National President's Night" on May 2 the Pittsburgh Section of the AIME offers an illustrated talk by

The Honorable Lewis E. Young
Consulting Mining Engineer
President AIME

Even more than Brutus, Dr. Young is an honorable man. So are all engineers, all honorable men.

We Are Threatened

One of our Communist members apparently does not share our expressed belief in the last issue that the North Atlantic Pact is desirable, for we have received an anonymous communication in big black print (no skull and crossbones however) that says: "North Atlantic Pact Means the Beganing of Wa #III and the End of America and You!" [sic] With only a slight tremble, we agree that our friend may be right, but the alternative, as we see it, is to let the Communists take over the world peaceably, and we prefer to take the chance of standing our ground with a gun.

Gold in the Black Hills

Robert J. Casey has just published a book entitled "The Black Hills and Their Incredible Characters," and those interested in the folklore of one of our oldest mining districts will find much in it of interest. It includes, of course, the story of the biggest gold mine in the United States, the Homestake, and of other attempts to mine gold in the district. "It seems strange to me that there should be so little successful mining of precious metals in these parts today," the author reports himself as saying to Chambers Kellar, general counsel for the Homestake. "Surely there is as much gold in the ground as was ever taken out." "Well," said Mr. Kellar, "that is one theory. The other theory is that there isn't."

The matter was never so clearly stated in an AIME T.P.

Among the Student Associates

This "rogues gallery" represents about half of the 100 members of the Mackay School of Mines' Crucible Club. President A. H. Barrios is seen at extreme left.

Lafayette College

The John Markle Society's last technical meeting of the year was held on April 26. It was highlighted by a spectacular demonstration of velocity power tools, given by W. S. McConnell, of the Mine Safety Appliances Co. A slide film, "The Changing World," was also presented, with a commentary by William Shaffer, metallurgical engineer of the class of '49. Society members wound up the session with the usual refreshments.—*John Siergiej, Jr., Secretary.*

Louisiana State University

The reactivated Student Chapter at LSU has been busy meeting bimonthly to hear interesting speakers, and arranging a student paper contest. Dan Grady, of the Hycalog Co., addressed the March 24 meeting, presenting a paper on "Hydrocarbon Profiles in Mud Logging." On April 7, J. E. Smith, of the Johnston Oil Field Service Corp., offered his paper entitled "Drill Stem Testing Procedure in Open Hole."

The Chapter is now engaged in a student paper competition, the winners of which will present their work before the Delta Section, AIME, sometime in April.—*Mahlon F. Manville, Corresponding Secretary.*

Yale University

At its Dec. 2 meeting, the Yale Metallurgical Society was addressed by Richard J. Christine, metallurgist of the American Brass Co. at Torrington, Conn. Mr. Christine discussed the question of what industry expects of the newly graduated metallurgist. The personal relations of the engineer in industry were emphasized, and Mr. Christine told some of his interesting experiences. Following the formal address questions were asked and discussion ensued. The February 15 meeting was addressed by I. J. Hook, research Engineer of American Brass Co., Ansonia, Conn. The Society's President, Denis A. Nankivell, introduced Mr. Hook, who discussed the general aspects of Anaconda copper production in the Montana area before showing a movie on this subject. The movie covered the mining, smelting, and refining of copper in the district. Mr. Hook concluded his address, and the meeting, with a discussion period. About twenty-five members attended the meeting.—*Voluntine T. Potter, Secretary.*

Current Institute Papers

The following papers in the three journals are available to Institute members. Each member will receive one classification in the journal of his choice. Members may obtain papers in the two remaining classifications by purchasing the indicated issue of the journal at $0.75 per copy. There are no separate copies of the individual papers. ME means Mining Engineering, JM Journal of Metals, and JPT Journal of Petroleum Technology.

Metals Papers

Cadmium Recovery Practice in Lead Smelting. By P. C. Feddersen and H. E. Lee. February JM.

Development of Muffle Furnaces for the Production of Zinc Oxide and Zinc at East Chicago, Ind. G. E. Johnson. February JM.

The Effect of Orientation Difference on Grain Boundary Energies. By. C. G. Dunn and F. Lionetti. February JM.

Solubility Relationships of the Refractory Monocarbides. By J. T. Norton and A. L. Mowry. February JM.

Pressure Distribution in Compacting Metal Powders. By Pol Duwez and Leo Zwell. February JM.

Preferred Orientation in Rolled and Recrystallized Beryllium. By A. Smigelskas and C. S. Barrett. February JM.

Self-Diffusion in Sintering of Metallic Particles. By G. C. Kuczynski. February JM.

Homogeneous Yielding of Carburized and Nitrided Single Iron Crystals. By A. N. Holden and J. H. Hollomon. February JM.

The Surface Tension of Solid Copper. By H. Udin, A. J. Shaler, and John Wulff. February JM.

The Ionic Nature of Metallurgical Slags—Simple Oxide Systems. By John Chipman and Lo-Ching Chang. February JM.

Properties of Chromium Boride and Sintered Chromium Boride. By S. J. Sindeband. February JM.

Magnesium-Lithium Base Alloys—Preparation, Fabrication, and General Characteristics. By J. H. Jackson, P. D. Frost, A. C. Loonam, L. W. Eastwood, and C. H. Lorig. February JM.

The Recovery of Cadmium from Cadmium-copper Precipitate By G. H. Anderson. March JM.

Electrolytic Zinc at Risdon, Tasmania, Major Changes Since 1936. By S. W. Ross. March JM.

The Hardenability Effect of Molybdenum. By J. M. Hodge, J. L. Giove, and R. G. Storm. March JM

A Method of Examination of Fine Metal Powder Particles with the Electron Microscope. By Laurence Delisle. March JM.

The Effect of Ferrite Grain Size on Notch Toughness. By J. M. Hodge, R. D. Manning, and H. M. Reichhold. March JM.

Recrystallization and Microstructure of Aluminum Killed Deep Drawing Steel. By R. L. Rickett, S. H. Kalin, and J. T. MacKenzie, Jr. March JM

Precipitation Phenomena in the Solid Solutions of Nitrogen and Carbon in Alpha Iron Below the Eutectoid Temperature. By L. J. Dijkstra. March JM.

Concentration of the SO_2 Content of Dwight-Lloyd Sintering Machine Gas by Recirculation. By W. S. Reid. April JM.

The Crystal Structure of Ni_4W. By E. Epremian and D. Harker. April JM.

Sulphur Equilibria Between Iron Blast Furnace Slags and Metal. By G. G. Hatch and John Chipman. April JM.

The Beryllium-iron System. By R. J. Teitel and Morris Cohen. April JM.

Discussion, Institute of Metals Division. May JM.

The Effect of High Copper Content on the Operation of a Lead Blast Furnace, and Treatment of the Copper and Lead Produced. By A. A. Collins. May JM.

What is Metallurgy? By John Chipman. June JM.

The Sigma Phase in Ternary Cr-Co-Fe and Cr-Co-Ni Alloys. By P. A. Beck and W. D. Manly. June JM.

Distillation of Zinc From Copper Base Alloys and Galvanizers Drosses. By F. F. Poland. June JM.

Cadmium Recovery Practice at the Donora Zinc Works. By G. T. Smith and R. C. Moyer. June JM.

The Statistical Nature of the Endurance Limit. By J. T. Ransom and R. F. Mehl. June JM.

Investigation of Temper Brittleness in Low-alloy Steels. By S. A. Herres and A. R. Elsea. June JM.

Oriented Arrangements of Thin Aluminum Films on Ionic Substrates. By T. N. Rhodin, Jr. June JM.

P-type and N-type Silicon and the Formation of the Photovoltaic Barrier in Silicon Ingots. By J. H. Scaff, H. C. Theuerer, and E. E. Schumacher. June JM.

Microstructures of Silicon Ingots. By W. G. Pfann and J. H. Scaff. June JM.

Mining Papers

Alluvial Tin Mining in Malaya. By A. D. Hughes. March ME.

Sampling of Coal for Float and Sink Tests. By A. L. Bailey and B. A. Landry. March ME.

Drilling Blastholes at the Holden Mine with Percussion Drills and Tungsten Carbide Bits. By E. A. Youngberg. March ME.

Aerial Photographic Contour Maps for Strip Mines. By George Hess and R. H. Swallow. April ME.

Drying of Low-rank Coals in the Entrained and Fluidized State. By V. F. Parry, J. B. Goodman, and E. O. Wagner April ME.

Coal Washing in Colorado and New Mexico. By J. D. Price and W. M. Bertholf. April ME.

The Rupp-Frantz Vibrating Filter. By W. M. Bertholf and J. D. Price. April ME.

Synthetic Liquid Fuels from Coal. By J. D. Doherty. April ME.

Drilling and Sampling Unconsolidated Materials. By L. W. Dupuy. May ME.

Effects of Rod Mill Speed at Tennessee Copper Co. By J. F. Myers and F. M. Lewis. May ME.

Temperature Compensation of Old Type Askania Magnetometers. By T. Koulomzine. May ME.

Municipal Water Needs vs. Coal Mine Stripping. By G. M. Dexter. May ME.

Correlation of the Performance Characteristics of Domestic Stoker Coals With Their Chemical and Petrographic Composition. By R. J. Helfinstine and G. H. Cady. May ME.

Ready-Made Heat From Coal. By D. W. Loucks. May ME.

Sinking With the Hydro-mucker at Mather "B" Shaft. By J. S. Westwater. June ME.

Diamond Drilling Quartz-feldspar Intergrowths. By L. C. Armstrong. June ME.

Safety Practices at the Crestmore Mine of the Riverside Cement Co. By R. H. Wightman and G. H. Adams. June ME.

Sintering Characteristics of Minus Sixty-five and Twenty Mesh Magnetite. By Alan Stanley and J. C. Mead. June ME.

Humphreys Spiral Concentration on Mesabi Range Ores. By W. E. Brown and L. J. Erck. June ME.

Formation and Properties of Single Crystals of Synthetic Rutile. By C. H. Moore, Jr. June ME.

Coal Washing in Washington, Oregon and Alaska. By M. R. Greer and H. F. Yancey. June ME.

A Study of Coal Classification and Its Application to the Coking Properties of Coal. By Michael Perch and C. C. Russell. June ME.

An Evaluation of the Performance of Thirty-three Residential Stoker Coals. By J. B. Purdy and H. W. Nelson. June ME.

Work of the U. S. Geological Survey on Coal and Coal Reserves. By Paul Averitt. June ME.

Petroleum Papers

Semi-automatic Power-Operated Drilling Machinery. By M. E. True and B. L. Stone. February JPT.

The Core Recorder. By Clark Millison. February JPT.

Capillary Pressures—Their Measurement Using Mercury and the Calculation of Permeability Therefrom. By W. R. Purcell. February JPT.

Effects of Transient Conditions in Gas Reservoirs. By D. T. MacRoberts. February JPT.

Outline of Weather and Wave Forecasting Techniques. By A. H. Glenn and J. E. Graham. February JPT.

A Study of Oil and Gas Conservation in the Pickton Field. By J. R. Welsh, R. E. Simpson, J. W. Smith, and C. S. Yust. March JPT.

Methane Hydrate at High Pressure. By Riki Kobayashi and D. L. Katz. March JPT.

Method of Establishing a Stabilized Backpressure Curve for Gas Wells Producing From Reservoirs of Extremely Low Permeability. By E. R. Haymaker, C. W. Binckley, and F. R. Burgess. March JPT.

Gas Hydrates of Carbon Dioxide-Methane Mixtures. By C. H. Unruh and D. L. Katz. March JPT.

Gas Hydrates of Carbon Dioxide-Methane Mixtures. By C. H. Unruh and D. L. Katz. April JPT.

Gulf of Mexico Floating Drilling Tender. By C. P. Besse and G. W. Osborne. April JPT.

Estimation of Reserves and Water Drive From Pressure and Production History. By E. R. Brownscombe and Francis Collins. April JPT.

Relation Between Pressure and Recovery in Long Core Water Floods. By J. N. Breston and R. V. Hughes. April JPT.

News of AIME Members

Harry L. Kirkpatrick

Alvey B. Rushton

George R. Eadie

Two more outstanding coal mining students, **Harry L. Kirkpatrick** of the University of Kentucky, and **Alvey B. Rushton,** of Lehigh University, have been honored by the Old Timers Club for scholastic excellence and evidence of being "likely to succeed." In addition, the Club has chosen **George R. Eadie** for citation as the most outstanding mining graduate of the year.

Mr. Kirkpatrick, who will receive his B.S. degree in June, was awarded an engraved gold watch at a general engineering assembly on Feb. 10 by **E. R. Price,** chairman of the AIME Coal Division. The lucky recipient is from Lexington, Ky., married, and the grandson of Harry L. Kirkpatrick, long prominent in the Western Kentucky coal industry.

Mr. Rushton hails from New Kensington, Pa., and is following in his family's footsteps in entering the coal industry. He's a veteran of two years' wartime service with the Marines, and was elected to Tau Beta Pi at Lehigh, which school he entered in 1944. **J. Ray Ulrich** made the gold watch award on March 24.

Mr. Eadie graduated from the University of Illinois in February of this year, and is now employed by the H. C. Frick Coal Co., Pittsburgh. His three years in the Army Air Forces followed a stint at the University of Southern Illinois, and then postwar studies brought him to the Missouri School of Mines, which he attended until 1948. Last summer he worked in the engineering department of the Mt. Olive and Staunton Coal Co., before returning to the University to take his degree. **Paul Weir,** president of the Old Timers Club, and head of the Paul Weir Co., Chicago, made the award, an engraved gold pocket watch, at the March 8 meeting of the Mineral Industries Society.

James Boyd, Director of the U. S. Bureau of Mines, will be the principal speaker at the Montana School of Mines Commencement exercises on June 10.

R. K. Comann in November of last year was transferred from the U. S. Gypsum Company's Plaster City, Calif., plant to the home office in Chicago. Since that time he has been in the engineering department and now is working at the Staten Island, N. Y., plant.

Walter B. Jones, state geologist of Alabama, was elected president for 1949-1950 of the Association of American State Geologists at its annual meeting in San Francisco. He succeeds **Garland Peyton,** director of the Georgia Department of Mines, Mining, and Geology.

Harry L. McKelvey has left La Luz Mines Ltd., Siuna, Nicaragua, to accept the post of mine superintendent with the New Idria Honduras Mining Co. at Santa Rosa de Copan, Honduras.

John A. Reeves, former University of Utah student, is assistant mining engineer for the Independent Coal and Coke Co., Kenilworth, Utah.

Jack W. Reinhart is a junior geologist for the American Smelting and Refining Co., Silver City, N. Mex.

A. E. Ring retired from active service as manager of the Southwestern division of the Western mining department of the American Smelting and Refining Co. on March 1. He is at home at 2001 E. Elm St., Tucson, Ariz.

K. A. Rogers has moved to Phoenix, Ariz., from Chippewa Falls, Wis. He had been with the Bureau of Mines and is now engineer with the Davis Dam Transmission System, Parker Dam, Calif.

James A. Ross has been elected first vice-president of the Industrial Diamond Association of America.

Morris E. Sandvig, University of Minnesota man, is a testing engineer for the Erie Mining Co., Aurora, Minn. His mailing address is Box 147, Aurora.

John T. Schimmel is time study engineer for the Consolidation Coal Co., Rivesville, W. Va., and can be reached at 108 Chicago St., Fairmont, W. Va.

Howard G. Schoenike is a geologist with the M. A. Hanna Co. at Iron River, Mich. He was previously employed as a geologist for Butler Brothers, who recently sold their interest to M. A. Hanna.

Dorner T. Schueler, Jr., has the job of engineer with the Industrial Minerals and Chemical Co., Berkeley, Calif., and is addressed in that city at 1155 Calusa Ave.

U. S. Smelting Staff Changes at Midvale

Robert Wallace retired on April 1 as manager of the Midvale plant of the U. S. Smelting Refining and Mining Co. after 33 years of continuous service with the Company, beginning on Jan. 13, 1916, when he was employed as general roaster foreman. By 1924 he was superintendent of the Midvale plant and in 1939 he became its manager. He now winds up 44 years of interesting work and achievement in the mining and smelting industry. After graduating from Harvard in 1905 he went to work in the engineering department of the Cleveland-Cliffs Iron Co., departing from Ishpeming for Old Mexico where he was mining engineer for the AS&R at their properties in Santa Eulalia and Sierra Mojada. Back in the States in 1909, he worked for the Mammoth Copper Co., and the American Zinc, Lead and Smelting Co. In 1914 he went to Utah to work at the Utah Apex in Bingham as a metallurgist in the concentrator. Mr. Wallace recalls that while he was metallurgist of the mill during 1914 and 1915 flotation machines were installed to extract lead minerals from the ore and this, he believes, was the first application of flotation to concentrate lead ores that was made in the State of Utah.

Hugo L. Johnson became general superintendent of the Midvale plant of the U. S. Smelting Refining and Mining Co. on April 1 and has direct charge of all operations of the plant and of the ore purchasing department formerly handled by Mr. Wallace. Mr. Johnson started with the Company 33 years ago as a draftsman in a subsidiary, the U. S. S. Lead Refinery. In 1918 he was made supervisor of the employe service department and four years later became superintendent of the silver refinery. In 1931 supervision of the engineering and maintenance department was added to his duties. He then transferred to Utah as superintendent of the gold and silver parting plant at Midvale which was built in 1933 along with the Parkes process lead refinery. Continuing his upward motion, he was in charge of the construction and operation of a pilot zinc plant, night superintendent at Midvale, assistant smelter superintendent, and in 1946 was made assistant to the manager of the plan, which position he held until his recent promotion.

Archie A. Nelson, who has been research engineer and who will continue to direct that work, has been appointed assistant smelter superintendent. He succeeds **W. M. Whitecotton,** who has been assistant smelter superintendent and in charge of blast furnace operations, and who requested that he be relieved of the responsibility of these posts because of his health. Mr. Whitecotton will remain at the plant as research engineer. Mr. Nelson has been with the Midvale plant for the past twenty years except for six years as mill superintendent and assistant manager of a company subsidiary, MicMac Mines Ltd. in Canada. Mr. Whitecotton was foreman of the cadmium plant of Midvale in 1915 and 1916, after which he was with the Electrolytic Zinc Co. of Australasia as superintendent of its zinc plant at Hobart, Tasmania, until 1922 when he returned to this country. After eleven years in various places, he returned to Midvale as roaster superintendent.

M. J. Belich has become general roaster foreman at Midvale. He has worked there since he finished his studies at the University of Utah on the 1939 W. H. Eardley scholarship in metallurgy awarded by the Company. Since 1946 he has been night superintendent.

John F. Schultze is a metallurgist in the Leadville milling unit of the American Smelting and Refining Co., Leadville, Colo. He receives his mail there at Box 27.

R. R. Schultz has resigned from his post with the Nordberg Mfg. Co. to become sales manager of the Willamette Iron and Steel Co., Portland, Oreg.

John E. Schumacher, Jr., although addressed at P. O. Box 689, Pottsville, Pa., is a field geologist for the Stanolind Oil and Gas Co., Casper, Wyo.

Henry E. Schweinsberg, former vice-president and sales manager of the Fields Mfg. Co., is president of CARBIT, Inc., Highland Bldg., Pittsburgh 6, Pa.

George C. Selfridge has been appointed mine superintendent of the Fresnillo and Zacatecas mines of the Fresnillo Co., Fresnillo, Zac., Mexico. He was formerly associated with the Manhattan District and the Atomic Energy Commission as chief of the exploration branch, raw materials division.

Paul Semko is chief of party of the geophysical exploration division of the Pennsylvania Turnpike Commission, doing geophysical field operations and earth resistivity determinations. His address is 646 Warren Ave., Kingston, Pa.

Arthur E. Shannon is sales engineer for the Jeffrey Mfg. Co., with a mailing address at P. O. Drawer 1078, Morgantown, W. Va.

Charles Gottschalk

Charles Gottschalk spent a month this spring in the States visiting his son and family in Baltimore after completing a two-year assignment in Japan as consultant on coal production problems. He is attached to the economic section of Military Government and expected to report back for duty early in May.

Ely Silverman, of the Tennessee Copper Co., is in Oak Ridge, Tenn., studying the techniques of using radioisotopes in research. He is one of 32 scientists from the States and several foreign countries enrolled in the seventh in a series of one-month courses offered by the Oak Ridge Institute of Nuclear Studies.

Raymond L. R. Slacke is employed by the National Coal Board, West Midlands division, in the capacity of area coal preparation engineer, Shropshire and South Staffordshire area, England.

R. Cecil Smart presented his paper on the Restoration of Productive Efficiency in Britain's Coal Industry before the Economic Reform Club and also before the Association of Colliery Managers, on Feb. 10. The paper's value was attested by the fact that a number of members of the National Coal Board attended his lecture, including representatives from the London office of the Board. His papers will be published in the proceedings of the National Association of Colliery Managers and quoted in the Iron and Coal Trade Review.

C. T. Smith, after 32 years as secretary and treasurer of the Missouri Co-

Mr. Mudd receiving citation for Egleston Medal from C. P. Davis, president of Columbia's Engineering School Alumni Association.

Harvey S. Mudd spoke on the importance of physical metallurgy in counteracting the depletion of our mineral resources at a meeting of the Columbia University Engineering School Alumni Association at which he was awarded the Egleston Medal for distinguished engineering achievement. He said in part:

"The second World War has given rise to much debate about the exhaustion of our mineral resources and in the name of conservation many measures have been proposed. That exhaustion is taking place cannot be denied. It is the rate of exhaustion that now gives concern, and properly so, but it should not be viewed with too great alarm.

"I cannot convince myself that the advance of civilization will be retarded by a deficiency of mineral resources, providing one condition is met, which condition is that research in the fundamental and applied sciences and engineering is wisely and generously supported. Our physical metallurgists are offering new alloys and new processes that give materials having superior properties to engineers with which to work and it is incumbent upon the engineers to make the best use of them, always with the future in mind."

In discussing his talk Mr. Mudd expressed concern over the relatively few students being graduated in metallurgical engineering as compared to other fields of engineering, and the need for metallurgical engineers in research and industry. He noted that research work required a particular type of individual with postgraduate training. Research work being done now in universities not only produced needed information but either directly, or through the faculty, helped to teach research techniques and helped universities to hold some exceptionally talented faculty that they could not otherwise hold.

He commented on the rapid industrial recovery being made in many of the western European countries. He was opposed to both direct subsidies and high tariffs, and pointed out that under normal conditions both were extravagant and interfered with free enterprise.

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balt Co. of Fredericktown, Mo., has resigned that post and now has retired from active business. Previous to joining Missouri Cobalt, he was accountant for El Oro Mining and Railway Co. at El Oro, Mexico. Mr. Smith can be reached at 118 E. Coolidge St., Harlingen, Texas.

Richard A. Smith retired as state geologist on the Michigan Geological Survey in 1946. Now he is special lecturer in geology at Michigan State College and is doing consulting work.

Robert E. Sorenson has been made chief engineer of the Hecla Mining Co. at Wallace, Idaho.

H. A. Steane has been appointed to the post of mill superintendent of Uruwira Minerals Ltd., Mpanda, via Uvinza. Tanganyika.

J. B. Stapler recently received appointment as managing director of Marsman Hongkong China, Ltd., P. O. Box 500, Hongkong & Shanghai Bank Bldg., Hongkong. Mr. Stapler has been with the Marsman interests for sixteen years and prior to the war was one of the senior engineers of Marsman & Co. The rapid development of mines such as Itogon, San Mauricio, and United Paracale, was largely due to his capable direction. Since his return from internment he has been a director of Marsman Investments, Ltd., and connected with the Marsman Co. of California directing from San Francisco the operations on Okinawa. Mr. and Mrs. Stapler departed on the President Cleveland March 9 for Hongkong where Mr. Stapler will take over his new duties.

Forest J. Sur, mining engineer and geologist formerly with the Bureau of Mines and now with Joe T. Juhan, recently completed a six-month examination and report on an oil shale project near Meeker, Colo., for Juhan and Associates. This done, he went to Farmington, N. Mex., to complete a reconnaissance of San Juan County, with the object of choosing sites for drilling deep wells to Pennsylvanian formations. He spent seven months on this work last year.

William H. Swayne, formerly in the geological department of the Anaconda Copper Mining Co., has gone to Santiago, Chile, as exploration geologist for Cia. Sud Americana Exploradora de Minas.

Ray E. Tower has resigned as general superintendent of the Mica Mountain Mines, South Pass City, Wyo. His new address is 621 S. W. Temple St., Salt Lake City, He is currently interested in mine development in the Masonic area, Mono County, Calif., and in the Tintic area in Utah.

Max A. Tuttle for the past 22 years has been employed by the Knox Consolidated Coal Corp. On Dec. 16, 1948, he became associated with the Enos Coal Mining Co., 1405 Merchants Bank Bldg., Indianapolis 4, Ind.

Gailen T. Vandel has accepted a post with the Cerro de Pasco Copper Corp. in Peru and expected to leave the States early in March.

Roy Van Zante, who was addressed at the Bureau of Mines in Rolla, Mo., can now be reached at the U. S. Vanadium Corp., Rifle, Colo.

L. A. Walker returned to Salt Lake City in March from a two-month professional trip to Colombia. Traveling to his destination in the Andes near the border of Ecuador his modes of transportation scaled from plane to auto and finally to mule. He stopped over long enough to meet members in Medellin, Cali, and Pasto.

S. Power Warren wishes to advise his friends that he has returned to Washington, D. C., after spending most of the first three months of the year studying the various mines and phosphate matrix treating plants of the

Florida field. During his stay he attended the meetings of the local phosphate group of the AIME and reports that the meetings were well attended by representatives of all the operating companies and the programs were enjoyable. His advice to the members contemplating a trip into this field is that they arrange their schedule to be there the first Monday of the month because at no other time will so many members be available in one place.

Donald R. Wells, of 153 Fallis Rd., Columbus 2, Ohio, is employed at Battelle Memorial Institute as a laboratory technician in the electron microscopy division.

Richard H. Wesley, with his two partners, after several years' study of the field, have gone into the consulting geological engineering business as Wesley, Bedford and Stickel, 2370 Penobscot Bldg., Detroit 26, Mich. Mr. Wesley writes: "We are in a position to carry on any type of exploration work any place in the world. This includes airborne geophysical work. Our laboratories are equipped for analyses of mining, petroleum products, and complete studies in soil mechanics. We can carry on spectrographic, petrographic, and core testing analyses. Our activities are not only confined to mining and petroleum, but also embrace preliminary studies for any major civil engineering project."

Joseph L. Walker, Jr., holds down the job of research mining engineer with the Rimersburg Coal Co., Rimersburg, Pa.

Robert Weagel, formerly shift foreman of the U. S. Section of the U. S. Smelting Refining and Mining Co., was appointed assistant general mine foreman of the section on Feb. 1.

J. P. Weir received the degree of bachelor of science in mining engineering at the midwinter commencement exercises of the Pennsylvania State College. At the same exercises, the technical degree, engineer of mines, was conferred upon his father, **Paul Weir.** Mr. Weir is now associated with his father in his consulting mining engineering firm, the Paul Weir Co.

Henry P. Whaley has the job of testing engineer with the Erie Mining Co., Hibbing, Minn.

A. D. Wilks was appointed superintendent of the Ordnance Factory, Katni (C.P.) India, over a year ago.

James W. Wivell has joined the staff at the Embarrass iron mine of Pickands Mather & Co.

Edward H. Snyder, president of the Combined Metals Reduction Co., Salt Lake City, was elected president of the American Zinc Institute at the annual

H. I. Young shows his successor as president, E. H. Snyder, a clock given him by the institute in token of his long service.

meeting of the board of directors, succeeding **Howard I. Young,** who has held the office since 1935. New vice-presidents include **George Mixter,** vice-president of the U. S. Smelting Refining and Mining Co., Boston, and **Raymond F. Orr,** president and general manager of the Athletic Mining and Smelting Co., Ft. Smith, Ark. **Erle V. Daveler,** vice-president of the American Zinc, Lead and Smelting Co., New York City, was elected treasurer and **Ernest V. Gent** was reelected executive vice-president and secretary. To serve as directors until 1952 are **A. E. Bendelari, O. W. Bilharz, C. M. Chapin, Jr.,** and **S. H. Levison.**

L. E. Young visited the McLeod River property of Canadian Collieries (Dunsmuir) Ltd., Nanaimo, B. C., the first week in April, thence proceeding to the Vancouver Island headquarters of the Company for an inspection of properties and consultation with **H. R. Plommer,** managing director.

• In the Metals Divisions

Paul H. Anderson received his M.S. degree on March 17 from the University of Minnesota, where he majored in metallography, and minored in industrial engineering. The title of his thesis is "Hot Dip Tinning of Low Carbon Steels." He intends to remain at the university to work toward a Ph.D. degree.

Bernhard Blumenthal, formerly research metallurgist with the American Platinum Works, Newark, N. J., has joined the staff of the Argonne National Laboratory, Chicago, Ill., as an associate metallurgist.

Chester L. Brown can now be reached at the production division of the development and research department of the Continental Oil Co., Ponca City, Okla.

D. S. Cameron has been made open hearth superintendent of the Wickwire Spencer Steel Division of the Colorado Fuel and Iron Corp. Buffalo plant.

Harry A. Campbell, executive engineer for Fredric Flader, Inc., manufacturer of gas turbines, receives mail at 341 Goundry St., Apt. 4, N. Tonawanda, N. Y.

Gregory J. Comstock, director of research of the powder metallurgy laboratory at Stevens Institute of Technology, left on April 5 by military transport for a month's stay in England, Germany, and Austria, on a commission for the Bureau of Ships of the U. S. Navy Department. The purpose of the visit was to discuss parallel research projects with British Governmental agencies. He visited the Edelstahlwerke and Verienegeste-Stahlwerke plants in Germany to study the current industrial application of chromizing. During the recent war, Professor Comstock served in Germany for the U. S. Joint Chiefs of Staff. Returning to this country, he was connected with the FEA and later with the U. S. Department of Commerce. Professor Comstock, on the Stevens staff since 1939, occupies the first chair of powder metallurgy established in any institution of learning in the States.

Minu N. Dastur has the job of metallurgical staff assistant with H. A. Brassert & Co., New York City.

Robert B. Day, now with the Linde Air Products Co., New York City, is in the Company's cylinder inspection and maintenance department as inspector for purchasing of pressure vessels.

Gerhard Derge and **Frederick N. Rhines** have been appointed to full professorships in the department of metallurgical engineering, college of engineering and science at Carnegie Institute of Technology effective Sept. 1. Both men joined the Carnegie staff in 1934. Dr. Derge, who graduated from Amherst and received his Ph.D. from Princeton, is a member of the Electric Furnace Steel Conference Committee of the Iron and Steel Division, AIME. Dr. Rhines, a Michigan and Yale graduate, is the Aluminum Co. of America professor of light metals and a two-time winner of the Institute of Metals award.

Denton T. Doll joined the staff of the chemistry and metallurgy division of the University of California Los Alamos Scientific Laboratory on April 11. He had been with the Brush Beryllium Co. in Cleveland.

Kenneth L. Erickson is employed as a chemist in process research by the Basic Reduction Co., Henderson, Nev.

Clyde R. Tipton, Jr.

Clyde R. Tipton, Jr., joined the staff of the chemistry and metallurgy division of the Los Alamos Scientific Laboratory, P. O. Box 1663, Los Alamos, N. Mex. He had been with Battelle Memorial Institute.

R. R. Estill has been made general superintendent of the fluorspar division of the U. S. Coal and Coke Co. at Mexico, Ky., replacing **K. A. Johnston,** who was transferred to the construction division at Lynch, Ky., on March 15.

J. Leon Gittlen is president and production manager of the Consumers Metal Corp., smelters and refiners of metals, Detroit.

Lawrence G. Glasglow, although still employed in the materials laboratories of Chance Vought Aircraft, is reached now at 1113 Canterbury Ct., Dallas, Texas, since the company has moved from Stratford to Dallas.

Richard A. Gorski is foreman relief trainee in the blast furnace division of the Gary works of the Carnegie-Illinois Steel Corp. His address is 4447 Grant St., Gary, Ind.

Charles H. Homer, Jr., formerly plant superintendent of the Beaumont Iron Works Co. at Beaumont, Texas, has been transferred to the Dunkirk, N. Y., plant of the parent organization, American Locomotive Co., Alco products division, where he assumes the duties of chief quality control engineer. The Dunkirk plant produces heat exchangers, pressure vessels, prefabricated pipe, large diameter pipe and other equipment for the petroleum, chemical, and allied industries. Mr. Homer's new mailing address is 353 Temple St., Fredonia, N. Y.

Carl H. Himmelman is a supervisory trainee with the Timken Roller Bearing Co. at Canton, Ohio.

Harry K. Ihrig, vice-president and director of laboratories of the Globe Steel Tubes Co., Milwaukee, Wis., received a citation for distinguished service in engineering from the College of Engineering of the University of Wisconsin at Madison on March 15.

Paul Linz resigned as a partner of Carl M. Loeb, Rhoades & Co. effective April 30, to become personal assistant to Dr. Mauricio Hochschild in all enterprises with which he is connected.

Roger D. Moeller joined the staff of the chemistry and metallugy division of the University of California Los Alamos Scientific Laboratory on March 21. He had been with the Sharples Corp. research laboratory in Philadelphia.

Claude R. Monlux graduated from Yale in February and is now employed in a training capacity by the Ohio Steel Foundry Co. of Springfield, Ohio.

Norman F. Tisdale

Norman F. Tisdale, chief metallurgical engineer of the Molybdenum Corp. of America since 1933, has been made manager of sales. A native Canadian and graduate of Queen's University, Kingston, Ont., he is well known in the iron and steel industry.

David P. Moore, of 68 Muskoka Rd., Grosse Pointe Farms 30, Mich., is metallurgist for the Hudson Motor Car Co., Detroit, Mich.

Thomas C. Nelson, Jr., is metallurgist for the General Electric Co. at the Hanford works in Richland, Wash.

William H. Nikola, who lives at 83 Highland Ave., Jersey City, N. J., is quality control engineer for the Westinghouse Electric Corp.

Leon V. Omelka is working as testing engineer for the Permanente Metals Corp., Trentwood, Wash. His mail goes to S. 4114 Ivory, Spokane, Wash.

Joseph R. Patterson, Jr., is at 468 Market St., Beaver, Pa., working for the Babcock and Wilcox Tube Co.

Burton C. Person, of 315 St. Louis St., Baton Rouge, La., is metallurgist in the Louisiana division of the Esso Standard Oil Co.

Fred P. Peters, editor-in-chief of *Materials & Methods,* has been made vice-president and publishing director of the book division of the Reinhold Publishing Corp., New York City 18.

Chester F. Robards is research metallurgical engineer on aircraft engines at the NACA Lewis Flight Propulsion Laboratory, Cleveland. He lives at 14029 Gramatan Ave., Cleveland 11, Ohio.

Leonard W. Rothe is a metallurgical trainee with the Armco Steel Corp., Middletown, Ohio. His home address is 2205 Lamberton St., Middletown.

Philip S. Savage, Jr., is working for the McCallum Bronze Co., Buffalo, N. Y. His address there is 242 4th St.

Arthur B. Shuck, formerly a research metallurgist with the metal products division of the Koppers Co., is now in the metallurgical section of the Argonne National Laboratory.

John A. Swift has formed the Swift Industrial Chemical Co., Hartford 1, Conn., to manufacture a complete line of chemical and metallurgical materials for metal cleaning, blackening, and heat treating. Mr. Swift has had extensive experience in the metal manufacturing, fabricating, and processing industries, having done work for the Western Electric Co., the American Steel and Wire Co., the Billings and Spencer Co., and the Heatbath Corp. In addition to his industrial experience, he has organized and instructed various metallurgical courses for evening divisions of the Hillyer College, New Haven Junior College, and the University of Connecticut. A registered metallurgical engineer in Connecticut, he is also engaged in consulting on heat treating, testing, surface finishing and treating, and fabricating of metals and metal products. Recently he served on the New England AIME Committee for the 1949 Regional Conference held in Springfield, Mass.

J. D. Swindale has left the Mond Nickel Co. and has resumed his war-interrupted studies at Birmingham University, Edglaston, Birmingham 15, England.

E. T. Turkdogan was recently awarded the degree of Master of Metallurgy in recognition of his research work on slag metal reactions at the University of Sheffield, Sheffield, England.

Joe G. Uzelac, formerly a student at the University of Utah, is working for the Denver Equipment Co., Denver, Colo., as a metallurgical engineer.

Ralph L. Wilcox, metallurgical engineer and vice-president of the Gerity-Michigan Corp., is now serving on loan from his company as chief of the nonferrous metals branch of the ECA in Washington, D. C. Mr. Wilcox says he would welcome a visit from any of his old WPB friends.

Tzuliang Wu is research engineer at the Institute of Industrial Research, Syracuse University, Syracuse, N. Y.

• In Petroleum Circles

Bernard W. Allen, former Colorado School of Mines student, has become a petroleum engineer with the Ohio Oil Co., Box 530, Cody, Wyo.

Joseph A. Buland, Jr., is currently employed by the Baroid Sales Division of the National Lead Co. as drilling mud service engineer in southwestern Wyoming. His mailing address is in care of the Teton Hotel, Riverton, Wyo.

John H. Dante is geologist and production manager with F. G. Fox and W. J. Fox of Owensboro, Ky. His address there is 829 Griffith Ave.

Harry Dereniuk, formerly with the Republic National Gas Co., is now with the Conroe Drilling Co., Corpus Christi, Texas, as a petroleum engineer.

Hal B. Fullerton, Jr., is an engineer in training with The Texas Co. at Long Beach, Calif. He lives at 336 Prospect Ave., Long Beach.

James R. Graham, after graduating from Louisiana State in January, accepted a job as petroleum engineer with the Standard Oil Co. of Texas. At present he is located in the Western division office in Midland, Texas.

Edward A. Gribi, Jr., finished his studies at Stanford last December and has been employed as a geologist by the Sinclair Petroleum Co. for exploration work in Ethiopia.

John J. Jakosky, Jr., is engineer with International Geophysics Inc., 1063 Gayley Ave., Los Angeles 24.

Robert Kamon, former University of Texas student, is a junior partner in the firm, Kamon Petroleum Exploration, Cisco, Texas.

W. D. Beach

Omaha Public Library
Reference Dept.

W. D. Beach, petroleum engineer with the Gulf Oil Corp., recently joined the reservoir section of the executive department in Houston, Texas. He was formerly with the Gulf production division of the Gulf Refining Co. at Laurel, Miss. Mr. Beach is a Penn State graduate and served as secretary-treasurer of the Mississippi Subsection of the Delta Section, AIME, for the last half of 1948.

Julie Anne Klovstad

Julie Anne Klovstad, formerly geologist with the Shell Oil Co. at Jackson, Miss., is now doing consulting work in eastern Colorado as well as working toward a Ph.D. degree at the University of Colorado. She may be reached at 1135 University Ave., Boulder.

Philip Kraft resigned as a director of the Magma Copper Co. last December 15.

Frank E. Long graduated from the University of Oklahoma with a degree in petroleum engineering and at present is employed by the Magnolia Petroleum Co. as an engineer trainee. His address is General Delivery, Falfurrias, Texas.

Charles T. McClure, formerly with the Signal Oil and Gas Co., has become associated with the Youngstown Sheet and Tube Co., Los Angeles, as a field engineer.

R. B. Luke McCullar, who is sales engineer for the Western Co., can be reached at Box 310, Midland, Texas.

Kenneth E. Meyer, petroleum engineer with the Cities Service Co., New York City, has been transferred to the Gulf Coast division. His new address is 1440 Mellie Esperson Bldg., Houston 2, Texas.

James I. Morris is working for the Pure Oil Co. as district production engineer. Mail reaches him at Box 700, Worland, Wyo.

Harold L. Patterson is in Delmita, Texas, working for the Sun Oil Co. as a petroleum engineer.

Warren D. Phelps, who had been at Texas A&M, is working for the Magnolia Petroleum Co., Box 792, Drumright, Okla.

Edward W. Racek, former University of Tulsa student, has the job of petroleum engineer for the Acme Well Supply Co., New York City. His home is at 36 Dogwood Lane, Levittown, Hicksville, N. Y.

Leslie B. Riggins, Jr., is chief engineer and geologist for the Portable Drilling Corp. His mail goes in care of the Company to P. O. Box 1318, Seminole, Okla.

John L. Ross is student engineer in the Gypsy division of the Gulf Oil Corp., Tulsa, but gets his mail at 112 N. Maiden St., Waynesburg, Pa.

E. S. Rugg can be reached in care of The Texas Co., New Castle, Wyo., where he is an engineer trainee.

Irl Sandidge, Jr., formerly with the Ohio Fuel Gas Co., Columbus, is now addressed at the United Fuel Gas Co., Box 1273, Charleston 25, W. Va.

W. S. Skinner received an M.S. degree in geology from Lehigh last June and during the summer worked for the New York State Science Survey to finish an investigation of oil possibilities in Otsego County, N. Y., which was begun in the summer of 1947. Last October he took the job of geologist with the South Penn Oil Co., Parkersburg, W. Va.

N. A. Tinker, Jr.

N. A. Tinker, Jr., joined the consulting engineering and geological firm of DeGolyer, MacNaughton & Murrell, Dallas, on March 15. While in Tulsa, before taking this latest post, he was sales contact engineer for the Weatherford Oil Tool Co., and prior to that was with the Gulf Oil Corp. as North Texas zone engineer.

Carle F. Sharp, Jr., formerly a student at the University of Texas, is working as a junior reservoir engineer for the Humble Oil and Refining Co. His mail goes to Box 357, Katy, Texas.

Duncan M. Smith, Jr., is now associated with the California Co. at the Lake St. John field at Waterproof, La.

Henry J. Stewart, now with the Union Oil Co. of California, P. O. Box 1365, Orcutt, Calif., used to be with the Sunray Oil Corp. in Alice, Texas.

Reid T. Stone is assistant field engineer for the Axelson Mfg. Co., Huntington Park, Calif. He lives in Apt. B, 8655 California Ave., Southgate, Calif.

H. J. Struth, former editor of the Petroleum Data Book, can now be reached in New York City at the American Petroleum Institute, Dept. of Information, 50 W. 50th St.

Ralph E. Taylor, who was with the Freeport Sulphur Co., joined the staff of Humble Oil and Refining Co. on March 15. He is addressed in care of the Company, Box 2180, Houston 1, Texas.

J. U. Teague resigned as superintendent of the Hogg Oil Co. on Feb. 1 to assume the presidency of the Columbia Drilling Co. This concern is engaged in the contract drilling of oil wells and maintains an office at 2011 Niels Esperson Bldg., Houston, with a field office and storage yard at West Columbia, Texas. Mr. Teague entered the oil business upon graduation from Rice Institute nineteen years ago as a petroleum engineer with the Humble Oil and Refining Co. He resigned in 1940 to become general superintendent of the Hogg Oil Co., interrupting this job with three years as a Lieutenant Commander in the Navy; he came out of service with seven battle stars in the Pacific theater. The Teagues live at 2406 Dorrington, Houston.

John F. Truchot, Jr., is working for the Rocky Mountain division of the Stanolind Oil and Gas Co. as a geologist. His address is P. O. Box 40, Casper, Wyo.

J. H. Van Zant is vice-president of the Van-Grisso Oil Co., 1414 Fort Worth National Bank Bldg., Fort Worth 2, Texas.

W. Frank West is a petroleum-engineer trainee with the Continental Oil Co., P. O. Box 451, Ventura, Calif.

Obituaries

Sydney Hobart Ball

AN APPRECIATION BY DONALD M. LIDDELL

Sydney H. Ball, a member of the Institute for 42 years, and Director from 1924 to 1927, died of a heart attack on Friday, March 25, at his home in New York City, aged 71 years and three months. He was active until the last, having been on professional work in British Columbia in December, when he had the first attack of his fatal illness.

Born in Chicago, he received in 1901 an A.B. degree from the University of Wisconsin, which awarded him a Ph.D. in 1910. He was one of the few who came into prominence in his youth. Having worked for the U. S. Geological Survey from 1903 to 1907, he was asked in the latter year to head an expedition to examine the concession of the Société International Forestiere et Miniere du Congo for the Ryan-Guggenheim group. Three years of work firmly established his reputation as an able field geologist and executive. It is probable that the Congo-Angola diamond field, an outgrowth of this expedition, is the richest known. From that time on, his services were in demand in every part of the world. From 1917 until his death he was associated with the firm of Rogers, Mayer and Ball, of which he was the last survivor.

He was three times decorated by the Belgian Government, first as Chevalier de l'Ordre Royal du Lion; second as Officier de l'Ordre Royal du Lion; and finally in January of last year, Commandeur de l'Ordre de Leopold II, the highest Belgian decoration that can be given a civilian.

For many years he had contributed the annual articles on gemstones to the Bureau of Mines Minerals Year Book, and had written for various trade publications. He was a member of the New York Jewelers' Publicity Committee and was largely responsible for the House of Jewelers at the New York World's Fair. A study that the writer found hugely entertaining embodied a long research into the use of gemstones by the Pre-Columbian Indians of the Americas.

In pure geology he made his mark by a study of Tertiary dikes and Pleistocene glacial deposits, which established the existence of two distinct ages in the so-called Front Range. He carried out the first accurate mapping of Death Valley, Calif., about 1907.

Known to the engineers and geologists of the world by reputation, his personality endeared him to his associates here. He was a charter member and later president of the Mining Club of New York City. He had been a member of the Mining and Metallurgical Society of America for 37 years and was its president in 1933. He was president in 1930 of the Society of Economic Geologists, and an honorary member of the Gemological Institute of America and of the Chemical, Mining and Metallurgical Society of South Africa. He was a member and one-time governor of the Explorers Club, and a director of the CRB educational fund.

One can give only a bare outline of his long and busy life. One cannot put on paper a proper appreciation of his serious conversation or of his impish delight in jokes on those he knew well. Nor can one express an adequate acknowledgment of his readiness to put his knowledge at the service of his friends.

ELWOOD BACHMAN (Member 1940), who was recently engaged in the electrification of the Utah Copper Company's Bingham property, and who had been associated with the General Electric Co. since 1916, died last Dec. 16. A man of many talents and interests, Mr. Bachman, when he received his E.E. degree from the University of Utah in 1916, already had three years of mine electrification, railway construction, and test work behind him. After joining General Electric he worked on mine in-

stallations, mill electrification, and electrolytic investigation projects, was at the same time in complete charge of electrical engineering at the Flin Flon plant of the Hudson Bay Mining and Smelting Co. in Hudson Bay.

LeGRANDE D. BARRETT (Member 1939), mining engineer, is dead. Mr. Barrett died at the age of 33 in South Africa, while acting as a Mormon missionary there. He had resigned, in 1946, his job as acting branch manager for the Salt Lake City office of the Independent Pneumatic Tool Co., in order to go on the mission. After receiving his B.S. in mining engineering from the University of Utah in 1941, Mr. Barrett worked for the Kennecott Copper Co., and then as an engineer for the New Park Mining Co., at Keetley, Utah, before taking his last professional job in 1944.

ARTHUR J. BOASE (Member 1936), former college professor and most recently manager of the Portland Cement Association's structural and technical bureau in Chicago, died recently. Before entering the University of Colorado in 1915, Mr. Boase had been superintendent of construction for the Boston Colorado Power Co. in Denver, and then assistant city engineer at Boulder. From 1920 to 1921 he studied at the University of Pennsylvania, and during those years and for three years thereafter was an instructor in civil engineering at that school. He then headed the department of civil engineering at Pennsylvania Military College for six years before joining the Portland Cement Association.

CHARLES EDWARD BRUFF (Member 1920), died in Berkeley, Calif., on Feb. 25 at the age of 72. He had been San Francisco representative for the Stevens Adamson Co. until his retirement several years ago.

It was 1895 when Mr. Bruff graduated from Purdue University with his degree in engineering, and for the ensuing fifteen years he was a drattsman for the L. C. Trent Co., assistant manager for the Bradley Engineering and Machinery Co., and a representative of the Power and Mining Machinery Co. in New York. Then he went on in 1910 to build the Chino mill in Hurley, N. Mex., and journeyed to Alaska to supervise construction of the Gastneau mill in Juneau, both assignments under the aegis of the D. C. Jackling engineering firm. After 1915 he was a partner in the Bradley, Bruff and Labarthe organization, actively engaged in the construction of the Alaska Juneau mill and the building of many domestic and foreign metallurgical plants.

EDWARD E. BUCBEE (Member 1905), nationally known for his work in mining and metallurgical engineering, and a professor emeritus at MIT, died suddenly in Boston, Mass., on Feb. 2, at the age of 72. Professor Bugbee graduated from MIT in 1900 and began his career as an instructor in that school's metallurgical laboratory the same year. In 1903 he was made an assistant professor at Iowa State College, but later returned to MIT, retiring as an associate professor of mining engineering and metallurgy in 1941.

LUCIUS A. BUTLER (Member 1920), died, at the age of 60, on Jan. 23. His last professional association had been with the U. S. Engineering Department in San Francisco. Mr. Butler was a graduate of the Michigan College of Mines, and spent the early years of his career with mining companies in Canada, Minnesota, and Michigan. He worked with the Inland Steel Co. for three years, and then became a project engineer with the Michigan State Highway department. In 1938 he joined the engineering corps of the War Department, working in his native Michigan until he moved to San Francisco.

FOREST L. CAMPBELL (Member 1940), chief engineer for the exploitation department, southern division, of Richfield Oil Co., died on Jan. 11 in Long Beach, Calif., of a heart attack. The 51 year old engineer's career began in 1918 with the San Dimas Co., in Mexico. He received an A.B. degree from the University of California in 1921, spent five years with the oil and gas division and State Mining Bureau of California, and joined the Richfield Co. in 1928, as a resident engineer. Mr. Campbell's chief interest was in petroleum, although mining geology and mining and milling also fell within the scope of his many abilities.

MARK G. CARHART (Member 1943), died on March 6 at the age of 66. Thus was ended an amazingly versatile career which created a man with broad managerial and engineering talents. At the turn of the century, Mr. Carhart was a young farm worker. A year later he was managing a lighting and water plant in Bartow, Fla.—and so it went through the first two decades—a continued eclectic interest in public power and private industry, from sales, production, and engineering viewpoints. The year 1930 found him managing the Venezuela Power Co. at Maracaibo. He was no stranger to Latin America, having served for the previous three years as chief engineer for the Westinghouse Corp. in Mexico City.

From 1931 to 1941 he represented Canadian mining operators in New York, developing finances and supervising hard rock mining operations in gold, silver, and copper ores. It has been said that Mr. Carhart's career was truly representative of the fast-disappearing tradition of our industrial leaders who made their own careers from the rich clay of experience.

Cassius I. Cook

AN APPRECIATION BY W. J. COULTER

In the passing of Cassius I. Cook, general manager of the Consolidated Coppermines at Kimberly, Nev., on Dec. 22, the mining fraternity lost a well known friend in the industry. For many years Mr. Cook had suffered from an asthmatic condition and his death was attributed to complications resulting from an acute condition.

Mr. Cook was born near Portland, Oreg., June 18, 1886. He received his early education in Boise, Idaho, and graduated from the University of Idaho in 1911. In college Mr. Cook took an active part in athletics and for three years played on the football team.

After graduating from the University of Idaho, he became superintendent of the Silver City Mining and Milling Co., at Silver City, Idaho. From Silver City he went to Alaska in 1915 and later to British Columbia where he was employed by the Granby Consolidated Mining, Smelting and Power Co. In 1927 Mr. Cook left Anyox, B. C., where he was assistant mine superintendent, to accept a position as mine superintendent of Consolidated Coppermines. During the depression period, from 1932 to 1935 when Coppermines was closed down, he operated a small gold property in Grass Valley, Calif. In 1935 he went to Silverton, Colo., where he was general superintendent of the Shenandoah Dives Mining Co. until 1940 at which time he returned to Kimberly, Nev., as general manager of the Consolidated Coppermines, which position he held at the time of his death.

Mr. Cook was well known and had many friends in the mining field. He was an active member of the American Mining Congress, AIME, Colorado Mining Association, a director of the Nevada Mine Operators Association and a member of the Ely Rotary Club. Mr. Cook was also a member of the Kappa Sigma Fraternity.

The news of Mr. Cook's death comes as a shock to his many friends in the mining industry and to all who knew him.

HERSCHEL E. DENTON (Member 1946), managing director of General Petroleums Ltd., and a partner in the firm of Denton and Spencer since 1941, died on March 30 in Calgary, Alta. He was 41. Mr. Denton had a B.S. in mechanical engineering, which he achieved by alternate years of work with the Casper Oil Co. of Wyoming and attendance at New Mexico A&M. By 1938, he was production superintendent for the Anglo-Canadian Oil Co. in Calgary, and three years later joined the two firms with which he was associated until his death.

Howard Drullard

AN APPRECIATION BY HARVEY G. BOSTICK

Our AIME Directory notes: Drullard, Howard (Member '18) Min. Engr., Min. Machinery, 1026 Folsom St., San Francisco, Calif. Howard was proud of the AIME and his San Francisco Section affiliation.

He passed away in San Francisco on Jan. 29, of a heart attack. He was born in Buffalo, N. Y., in 1883, and attended private schools. He had a desire for mining and so became a shifter in the Second St. tunnel in Los Angeles and in the Elizabeth Lake tunnel of the Los Angeles water department. He contracted in the Ray Con. in Arizona, and subsequently managed the Hardenburg and Poundstone mines in California's Mother Lode. He also mined in Mexico and Central America.

While still a young man Howard was the victim of a blasting accident. Fortunately he pulled through with half vision in one eye. He never accepted this as handicap and this was evidenced by the fact that during the 1915 San Francisco exposition and fair, he represented five companies. Denver Rock Drill Co. was one of them, and after the closing of the Fair, Howard became affiliated with it. He was in turn Butte, Mont., manager, Salt Lake City manager, and then a representative for Gardner-Denver in Japan and Korea. Returning to the States in 1928, he attended Stanford University. In 1932 he decided to enter private business and established the firm which he managed until his demise.

This is an example of Howard's great human kindness and understanding: During his residence in Butte he became acquainted with Stephen Breckenridge, a miner totally blinded by an explosion. Encouraged by Howard to "carry on," Steve sold newspapers. Later, Steve moved to Los Angeles and has been on the corner of Fifth and Broadway for many years—selling newspapers—and is independent. Howard never failed to visit Steve during his Los Angeles calls.

Howard Drullard's counsel was kindly. His many, many friends all over the world will be saddened by his passing, but he was the type of man that will live on in memory and in spirit.

CECIL DRAKE (Member 1931), died in Taft, Calif., on Jan. 31, at the age of 49. He had worked for the Standard Oil Co. of California since 1946. With an E.M. in geology from the Colorado School of Mines, Mr. Drake went to Mexico in 1922 to work for the Eagle Petroleum Co. of Mexico, a Shell company, and remained there for eight years. Then he returned to the States to work for the Shell organization in Illinois and Oklahoma before joining Standard Oil.

Donald R. Knowlton

DONALD R. KNOWLTON (Member 1932), manager of the foreign department of the Phillips Petroleum Co., and perhaps best known for his work as director of production with the Petroleum Administration for War, died in Oklahoma City on March 9 after a short illness. He was 54. After taking his E.M. degree from Stanford University in 1924, Mr. Knowlton worked as an engineer for the Continental Oil Co. in Texas, joined the Phillips organization in 1930, becoming chief production engineer a year later. He recently stirred the business world with a magazine article on "The Semantics of Financial Reports" which censured the obscure, technical language in which financial reports are phrased.

LIONEL H. DUSCHAK (Member, 1917), professor of metallurgy at the University of California, died on Nov. 27. Born in 1882, Dr. Duschak received his A.B. degree from the University of Michigan in 1904, went on to take an M.A. and Ph.D. from Princeton four years later. With his degrees in chemistry, he taught that subject at Princeton from 1907 to 1909, then served four years as a research engineer for the Corning Glass Works and the next five years as a chemical engineer with the Bureau of Mines. Since 1921 he had been a consulting engineer, and in 1938 joined the faculty of the University of California.

CHARLES DENHAM GRIER (Member 1915), died on Dec. 2 in Johannesburg, South Africa, where he had been, since 1923, managing director of South African Cyanamid Ltd. Born in 1889 in Denver, Colo., Mr. Grier was a graduate of the Colorado School of Mines, also had an M.Sc. degree from the University of Washington. His early experience took him through the West, and then up to Alaska. The first World War found him a Second Lieutenant in the engineers. In 1922 Mr. Grier joined the American Cyanamid Co., and the following year took up residence in South Africa to manage that Company's interests there. In 1937 he took over the position he held at the time of his death. Although his main efforts dealt with cyanide production, Mr. Grier took an active interest in the whole mining and industrial development of southern Africa.

HORACE S. GULICK (Member 1916), whose career had taken him from Oklahoma A&M to the first vice-presidency of the New York Air Brake Co., died on Jan. 31 at the age of 63. Mr. Gulick received his B.S. degree from the above-named school in 1903, and within ten years had become chief chemist for the American Steel Foundries in East St. Louis, Ill. Later he was superintendent of testing and inspection at the More-Jones Brass and Metal Co., and in 1921 superintended the Ohio Brass Company's Mansfield, Ohio, plant. Six years later he assumed a similar position with New York Air Brake, remaining with the Company until his death.

ELWIN B. HALL (Member 1914), petroleum engineer and oil geologist, died in Los Angeles on Jan. 20. Mr. Hall was 57 and had been ill for three weeks prior to his death. At the time of his death he headed the E. B. Hall Co., which operated the Union Pacific oil properties at Wilmington, California. Mr. Hall was a graduate of the School of Geology of Stanford University, where he had been a three year letterman in rugby and notably proficient in other sports. He worked as a petroleum geologist for many years, and in 1937 became president of the Hall-Baker Co. in Los Angeles.

PHILIP H. HOLDSWORTH (Member 1938), for many years prominent in Oregon mining, died at his home in Seattle last Nov. 21. He was born in 1881. Back in 1908 Mr. Holdsworth was assistant superintendent for the Alaska Smelting and Refining Co. in Hadley, later went on to do mining and metallurgical work in California, Oregon, and the State of Washington. Since 1920 he had been a consulting engineer with offices in Seattle, was greatly interested in geophysical work, and made several investigations in both Oregon and Washington employing resistivity methods.

FREDERIC W. JAMES (Member 1917), died last Dec. 28 at his home in Belleville, N. J., at the age of 68. He was a vice-president and engineer at the N. P. Nelson Iron Works, Inc., in Passaic, and had been associated with the firm for 27 years. Born in St. Marys, Reading, England, he came to this country at an early age, attended Trinity School, and the Columbia School of Mines. From the

Necrology

Date Elected	Name	Date of Death
1948	E. Paul Keuper	December 1948
1916	H. H. Stout	April 18, 1949
1914	A. M. Tweedy	January 1949

turn of the century until he joined the Nelson organization, Mr. James had worked for nine engineering concerns, as both draftsman and engineer. He devoted twenty years to serving the Boy Scouts in his hometown, was also a member of the National Historical Society, and a frequent contributor to various engineering publications.

OTTO G. LEICHLITER (Member 1935), president of the Mudge Oil Co. of Texas, died last Sept. 8, at the age of 63. Although his professional education in mechanical and civil engineering was not acquired formally, but in home studies, Mr. Leichliter was, at the age of 22, superintending a tunneling operation for the Gold Cup Mining and Milling Co. in Colorado. Two years later he joined the Edward W. Mudge Co., becoming general manager of four affiliated companies, and building coal mines, coke ovens, blast furnaces, and developing oil properties. By 1934 he was vice-president of the Weirton Coal Co. and Union Electric Steel Co., and also held the position he retained until the time of his death.

R. JOHN LEMMON (Member 1930), died in Nenagh, Tipperary, Eire, on Jan. 26. A British subject, born in Sussex in 1878, Mr. Lemmon traveled far and wide in pursuit of his metallurgical career. He was a chemist on the Gold Coast in 1911, at work in Korea eleven years later, and within the next seven years had been a metallurgist in Brazil, Turkey, Russia, and Bolivia. In 1938 he joined Imperial Chemical Industries, Ltd., in Liverpool, and at the time of his death was a metallurgist for the Irish Exploration Co. in Tipperary.

NORMAN MACKENZIE (Member 1943), owner of the Coleville Engineering Co., Spokane, Wash., died Dec. 8 at the age of 53. Born in Canada, Mr. Mackenzie attended Manitoba College and Manitoba University, receiving his B.Sc. degree in 1919. His education was interrupted from 1914 to 1919 while he served as Lieutenant-Captain in the Canadian Engineers during World War I, leading a tunneling company in England, France, Belgium, and Germany. Varied engineering experience in this country and Canada brought him to the formation of his own company in 1936. He engineered and designed 24 rural electrification systems, carried on extensive electrical, municipal, and mining consultation activities until the time of his death. Mr. Mackenzie was also an active member of numerous professional and social groups, including the American Legion, BPOE, and the Spokane Press Club.

F. A. G. MAXWELL (Member 1936), who spent his whole professional career in the South African gold mining industry, died in Johannesburg on Dec. 26. He was 74 years old. Mr. Maxwell, a British citizen, was born in Freemont, Neb., received his education at the Colorado School of Mines, and went to South Africa immediately after graduating in 1895. He played a large part in the foundation and development of metallurgical practice on the Rand, and was consulting metallurgist to the General Mining and Finance Corp. for the past thirty-three years.

Arthur S. Nichols

ARTHUR S. NICHOLS (Member 1936), vice-president and sales manager of the Illinois Clay Products Co. for more than 23 years, died on Feb. 18. He was 48 years old. Mr. Nichols, chairman of the AIME Committee on Ceramic Materials, and chairman of the Chicago Section in 1944, was a native Chicagoan, educated at the University of Illinois. He began his career in 1922 with the Harbison-Walker Refractories Co., then became a technical adviser to the Chicago Firebrick Co. before joining the Illinois organization.

ALEXANDER G. McGREGOR (Member 1915), died on March 4 at the age of 69. Mr. McGregor had been in London for the past twenty-two years, serving as a consulting mechanical engineer to the Chester Beatty group of mining companies. To that group, he brought a formidable reputation as a designer and builder of smelting works, mining, and metallurgical plants. Some of the plants he designed and constructed include: the International Smelting Co. plant at Miami, Ariz.; the Cerro de Pasco Copper Corp's. smelting works in Peru; the Phelps-Dodge smelting units at Douglas, Ariz.; and the Inspiration C. C. Company's leaching plant. During the recent war he was a consulting engineer to the British Ministry of Aircraft Production.

Mr. McGregor had a B.S. from the University of Montana, was the author of numerous economic articles and books, among them "The Correct Economy for the Machine Age," "Collective Bargaining and Decadence," and "End Wages Conflict."

WALTER B. MILLER (Member 1916), electrical and mining engineer of Huntington, W. Va., died last September. Mr. Miller was born in 1878, attended West Virginia University and the University of Kentucky, and spent the bulk of his early career with Kentucky and West Virginia coal companies, except for a stint with the Army engineers during World War I. More recently he had been general manager for the Kentucky King Coal Co., and his last affiliation was with the Guyen Eagle Coal Co. in Huntington.

THOMAS MacDONALD ROBERTSON (Member 1948), died last Aug. 16 at the age of 23. A native of Glasgow, Scotland, Mr. Robertson graduated from the Royal Technical College in 1945, became a research metallurgist and later assistant melting shop manager in the Clydebridge steel works. He was an associate member of the West of Scotland Iron and Steel Institute, and a student member of the Institute of Metals.

THEODORE W. ROBINSON (Member 1884), who was vice-president of the Illinois Steel Co. until 1932, died on Dec. 30. Mr. Robinson had been a member and director of various public and private organizations, including positions as trustee of Northwestern University, a director of the City National Bank and Trust Co., the Illinois Bell Telephone Co., and chairman of Ditto, Inc. He was an MIT graduate, class of 1884.

WILLIAM A. ROSE (Member 1940), well known figure in the iron mining industry, died in Santa Monica, Calif., at the age of 66. Mr. Rose had been with the Pickands-Mather Co. for nearly a half-century. Born in Rochester, he received a degree in mining engineering from the University of Minnesota in 1906, became chief engineer for the Oilver Iron Mining Co., and then went to Pickands-Mather in 1913. He was appointed chief engineer there, and later assistant general manager. After a short stint in Cleveland for the Company, he retired in January 1947, and went to live in California.

A. D. SHANKLAND (Member 1917), metallurgical engineer on the staff of the operating vice-president of Bethlehem Steel Co., died Jan. 5. Of Mr. Shankland's 57 years, 35 were spent at Bethlehem. He first joined the Company as a chemist in 1914, upon graduation from Virginia Polytechnic Institute. Subsequently, he rose through the positions of metallurgy inspector, engineer of tests, and assistant general manager, to the post he was holding at the time of his death.

FREDERICK WARREN SNOW (Member 1917), president of the Vermont Copper Co., died in Hanover, N. H., on March 7. Mr. Snow had been with the Company since 1943, having served previously as general manager of the U. S. Mining and Smelting Co's. Mic-Mac mines in Quebec, and in the same capacity for the Eustis mine in Sherbrook, Que. A graduate of the University of Utah, Mr. Snow, had also spent thirteen years with the Magma Copper Co. in Superior, Ariz., and done mining and consulting work in the West and South America.

JAMES W. STARKS (Member 1945), director of the Illinois State Mines and Minerals Department, died in Springfield on Feb. 3, just seventeen days after he had taken over the post. Union members and coal operators alike had both heartily approved of his appointment last December. For the past 27 years he had been with the Peabody Coal Co., and for eight years was a mine officer in the department he came to head. He had also served four years as mine inspector in eight different counties, bringing a background of twenty-five years' varied experience to his work for the State.

DAVID R. THOMAS (Member 1904), Canadian mining engineer, died on March 4 at the age of 73, in Toronto. The British-born Mr. Thomas received his education and early training in his homeland, came to Canada in 1899 as an engineer for the Canadian Pacific Railway. Later he spent over ten years as a mine executive in Mexico, and another decade in mining, consulting, and managerial positions in Mexico, New York, and Canada, respectively. When he retired from active practice in 1930 he was a mining consultant in Toronto.

WILLIAM G. WAGNER (Member 1920), died in London, England, on March 19 at the age of 75. Mr. Wagner was managing director of George T. Holloway and Co., of London, which firm he had been with for the past forty-one years.

BAILEY WILLIS (Member 1892), seismologist and geologist of international fame, died in Palo Alto, Calif., on Feb. 19. Dr. Willis, who had made a lifelong study of earthquakes, was also a writer, lecturer, explorer, and world traveler, and extremely active until his ninety-first, and last year. A Columbia School of Mines graduate, he began his career as a mining engineer, later worked with the USGS, did research in China for the Carnegie Institution, and exploration in Argentine. In 1915 he took over Stanford University's department of geology, was forced to retire at sixty-five, but stayed on as professor emeritus until the time of his death. Just before his death he completed the second part of his two-volume biography, entitled "Friendly China."

Proposed for Membership

Total AIME membership on April 30, 1949, was 15,566; in addition 4187 Student Associates were enrolled.

ADMISSIONS COMMITTEE

James L. Head, Chairman; Albert J. Phillips, Vice-Chairman; George B. Cortese, T. B. Counselman, Ivan A. Given, George C. Heikes, Richard D. Mollison, and Philip D. Wilson.

Institute members are urged to review this list as soon as the issue is received and immediately to wire the Secretary's office, night message collect, if objection is offered to the admission of any applicant. Details of the objection should follow by air mail. The Institute desires to extend its privileges to every person to whom it can be of service but does not desire to admit persons unless they are qualified.

In the following list C/S means change of status: R, reinstatement; M, Member; J, Junior Member; AM, Associate Member; S, Student Associate; F, Junior Foreign Affiliate.

ALABAMA

Birmingham — SPARF, CHARLES ROY. (R,C/S—S-AM). Explosive serviceman, Hercules Powder Co.

ARIZONA

Klondyke—WILSON, GLEN BAYHA. (C/S—J-M). Mine superintendent, Athletic Mining Co.

Morenci—JACOBY, DWIGHT CLARK. (C/S—S-J). Hydrographic engineer, Phelps Dodge Corp., Morenci branch.

CALIFORNIA

Arcadia — DRAPER, DARRELL WELLINGTON. (M). Process service representative, Linde Air Products Co.

Bakersfield — CHRISTENSEN, JOHN ALEXANDER. (M). Geologist, British-American Oil Producing Co.

Berkeley—McCLELLAND, THOMAS HUTCHISON. (R,C/S—AM-M). Field manager, Asnazú Gold Dredging, Ltd.

Fontana — SAUSSAMAN, JOHN DANIEL. (M). Superintendent, blast furnace dept., Kaiser Co.

Los Angeles — TIPTON, KENNETH AMESTOY. (C/S—S-J). Insurance agent and broker, Kenneth E. Tipton Insurance Agency.

Maywood — BOWDEN, CARL B. (C/S—S-J). Assistant petroleum engineer, Union Oil Co. of California.

Pittsburg — CALDWELL, THOMAS EDGERTON. (M). Works metallurgist, Pittsburg works, Columbia Steel Co.

COLORADO

Cherry Hills Village — NICHOLSON, JAMES DOUGLASS. (M). General manager, The Mine & Smelter Supply Co.

Leadville — METZGER, GUINN EDWARD. (C/S—S-J). Junior metallurgist, Resurrection Mining Co.

Rangely — LEUZINGER, LESLIE ADOLPH. (C/S—S-J). Field engineer, Union Pacific Railroad Co. oil development, c/o The Texas Co.

DISTRICT OF COLUMBIA

Washington — BRABANT, PAUL HENRI. (M). Consulting engineer on Chinese mines, Pierce Management, Inc.

IDAHO

Wallace—HAALAND, OTTO ELMER. (C/S—AM-M). Mining engineer, Spokane-Idaho Mining Co. HAMILTON, WOODMAN CLARK, JR. (J). Shift boss, Day Mines, Inc.

ILLINOIS

Chicago — BIDDLE, GEORGE JOSEPH. (M). Assistant sales manager, Illinois Clay Products Co. JAIN, PADAM PRASAD. (M). Engineering supervisor, Killium & Co. SIREL, I. ALEXIS. (M). Metallurgist, Youngstown Sheet & Tube Co. SLOMER, JOSEPH J. (M). Supervisor, design-hydraulics, Goodman Mfg. Co.

Danville—MERLE, JAMES JOSEPH. (C/S—J-M). Assistant preparation manager, Ayrshire Collieries Corp.

Gillespie — HUBBART, CURTIS QUENTIN. (R,C/S—S-M). Mining engineer, Superior Coal Co.

Oakwood—MULLINS, RICHARD A. (M). Chief chemist, Ayrshire Collieries Corp.

Peoria — ANDERSON, ROBERT GRANT PHILLIP. (C/S—S-J). College graduate trainee, Caterpillar Tractor Co.

Salem—LOVE, RICHARD WILLIAM. (M). District petroleum engineer, The Texas Co. MULLINS, JESSE FRANK. (J). Petroleum engineer trainee, The Texas Co.

INDIANA

Bloomington — CALLAGHAN, EUGENE. (R,C/S—J-M). Professor of economic geology, Indiana University.

West Lafayette—HOPPE, ALBERT WILLIAM. (J). Instructor of metallurgy, Purdue University.

KENTUCKY

Benham—NEWMAN, ELMER KIMBER. (R-M). Chief engineer, International Harvester Co.

LOUISIANA

Minden — LARSON, KENNETH HENRY. (C/S—S-J). Junior computer, Stanolind Oil & Gas Co.

MASSACHUSETTS

Roslindale—PAYNTON, WILLIAM C. (C/S—S-J). Staff member of metallurgy project, MIT.

MICHIGAN

Detroit — STICKEL, JOHN FREDERICK, JR. (C/S—S-J). Geological engineer, Wesley, Bedford & Stickel.

Marquette—DeHAAS, GORDON TIFFANY. (C/S—S-J). Sales engineer, C. T. DeHaas Co.

MINNESOTA

Hibbing — McDERMOTT, WILLIAM FRANCIS. (J). Mechanical engineer, W. S. Moore Co.

Parkville—VAN EVERA, JAMES W., JR. (C/S—J-M). Chief mine engineer, Inter-State Iron Co.

MISSISSIPPI

McComb — MARTIN, WALTER OBIER. (C/S—S-M). Reservoir engineer, The Texas Co.

MISSOURI

Kirkwood — MASSMANN, ALBERT P. (M). In charge of preparation, Sinclair Coal Co. QUINN, MAURICE L. (M). Electrical engineer, Sinclair Coal Co.

St. Louis—SCALLY, JOSEPH EDWARD. (R,C/S—J-M). Vice-president, Harmony Oil Corp.

MONTANA

Cut Bank—FLUGSTAD, JOHN BUTLER. (RC/S—S-M). District engineer, The Texas Co.

NEW JERSEY

North Plainfield—COOK, CHARLES ROSCOE, JR. (C/S—S-J). Research metallurgist, Titanium Division, National Lead Co.

NEW MEXICO

Buckeye—DENSON, JACK McVAY. (C/S—S-J). Petroleum engineer trainee, The Texas Co.

Carlsbad—BINSWANGER, TOLSTOY LEO. (M). Extra supervisor, U. S. Potash Co.

Hobbs—BEARLY, MARION FRANK. (J). Student engineer, Gulf Oil Corp.

NEW YORK

Albany—BROUGHTON, JOHN GERARD. (C/S—J-M). Acting state geologist, N. Y. State Science Service.

New York — CULLITY, BERNARD DENNIS. (C/S—J-M). Scientific liaison officer, Office of Naval Research, American Embassy. JAMES, HAROLD FREDERICK. (M). Metallurgical engineer, Intsel Metals Corp. RAIKLEN, HYMAN ISAAC. (C/S—S-AM). Metallurgist, Wright Aeronautical Corp. VASSILOPOULOS, GEORGE. (C/S—S-J). Metallurgist, Wright Aeronautical Corp.

Syracuse—WU, TZU LIANG. (C/S—S-AM). Research engineer, Syracuse University.

White Plains — MULLANE, JOHN JOSEPH. (M). Production executive assistant, Standard Oil Co. (N. J.).

Yonkers—LESZYNSKI, WERNER J. (M). Editor, *Powder Metallurgy Bulletin*; member, Research staff, American Electro Metal Corp.

OHIO

Columbus—NELSON, HOWARD EDWARD. (J). Sales engineer, Jeffrey Mfg. Co.

Worthington — HALL, ALBERT M. (C/S—J-M). Metallographer-in-charge, Battelle Memorial Institute.

OKLAHOMA

Bartlesville—ALLEN, ERNEST, JR. (C/S—J-M). Reservoir engineer, Phillips Petroleum Co.

Ponca City — MESSER, ELMER S. (M). Research engineer, Continental Oil Co.

Tulsa—WHALEY, NEWTON PHILIP. (M). Consultant, Whaley Engineering Co.

OREGON

Salem—SCOTT, LEWIS EUGENE. (J). Geologist, State Highway Dept.

PENNSYLVANIA

Bethlehem—LARSON, HUGO CHRISTIAN. (M). Metallurgical engineer and secretary of Bethlehem plants committees, Bethlehem Steel Co.

Norristown — MATHEWS, DONALD R. (M). Open-hearth superintendent, Alan Wood Steel Co.

Pittsburgh — EICHELMAN, GEORGE HENRY, JR. (C/S—S-J). Intermediate metallurgist, Westinghouse Research Laboratories. SEWARD, JOHN EMRIE. (C/S—S-J). Practice engineer, Jones & Laughlin Steel Co.

State College — MELDRUM, ALAN HAYWARD. (C/S—S-M). Research assistant, Penn. State College.

SOUTH DAKOTA

Belle Fourche — ELLERMAN, OTTO MATHIAS. (R,C/S—J-M). Local manager, Wyodak Chemical Co.

TEXAS

Athens — HARGRAVE, AUSTIN JESSE. (M). District production manager, Lone Star Gas Co.

Greggton—SCHMIDT, LOREN ARNOLD. (J). Junior petroleum engineer, Stanolind Oil & Gas Co.

Houston—KINDSVATER, PAUL A. (R,C/S—S-M). District engineer, British-American Oil Producing Co. LUGOL, GILBERT M. (J). Ingenieur des Mines en Service Detache, Direction des Carburants. MENETRIER, CLAUDE F. (J). Drilling engineer, Regie Autonome des Petroles. ZWICKY, ROBERT WILLIAM. (C/S—S-J). Junior exploitation engineer, Shell Oil Co.

Kilgore—GEFFS, CHARLES RALPH. (R,C/S—S-J). Petroleum engineer, Sun Oil Co.

Longview — OKERSON, STANLEY JOHN. (J). Equipment engineer, Stanolind Oil & Gas Co.

Lubbock — CHAPIN, ESROM LON. (M). Petroleum engineer, Stanolind Oil & Gas Co. COTEY, BRADFORD J. (M). Manager, Oilwell Chemical Service Co. CROSBY, GEORGE ERNEST. (R,C/S—S-M). District engineer, Stanolind Oil & Gas Co.

Midland — DARDEN, FRANK. JR. (J). Junior petroleum engineer, Humble Oil & Refining Co. GILL, PERRY A. (M). Division superintendent, Sunray Oil Corp. MOORE, WILLIAM DAVID. (C/S—S-J). Junior petroleum engineer, Humble Oil & Refining Co. PIERCE, EDWARD JARVIS. (M). Petroleum engineer, Mid-Continent Petroleum Corp.

Odessa—MASSEY, JOHN KILBANE. (AM). General superintendent, Plastic Process Corp. PRUITT, LEE EUGENE. (M). District engineer, Sunray Oil Corp.

Pampa — THOMAS, FRANK GRAHAM. (M). District petroleum engineer, Humble Oil & Refining Co.

Snyder—SHOR, GEORGE, JR. (C/S—S-J). Seismograph party chief, Seismic Explorations, Inc.

UTAH

Midvale—SCHOLL, LESTER KEITH, (C/S—J-M). Metallurgist, U. S. Smelting Refining & Mining Co.

Salt Lake City — CROPPER, KENNETH FAY. (R,C/S—J-M). Engineer, U. S. mine, U. S. Smelting Refining and Mining Co. GODBE, MURRAY CHARLES. (C/S—S-J). Geological draftsman, American Smelting & Refining Co.

Sunnydale — HEERS, ROBERT G. (C/S—J-M). Coal mine manager, Kaiser Co.

WASHINGTON

Richland — NELSON, THOMAS C., JR. (C/S—S-J). Metallurgist, General Electric Co.

WEST VIRGINIA

Morgantown—HESS, ROBERT HAROLD. (C/S—S-J). Assistant mineralogist, West Virginia Geological Survey.

Rensford — McCORMACK, GEORGE EDWARD. (C/S—S-J). Engineer, Hatfield-Campbell Creek Coal Co.

WYOMING

Cody — EVANS, RICHARD LEWIS. (M). District reservoir engineer, Stanolind Oil & Gas Co.

Worland—MORRIS, JAMES IRVING. (C/S—S-J). District production engineer, Pure Oil Co.

ALASKA

College — MONROE, D. GILBERT. (C/S—S-J). Junior engineer, Boeing Aircraft Co.

BRITISH COLUMBIA

Vancouver—BARER, RALPH DAVID. (C/S—S-J). Instructor in physical metallurgy, University of British Columbia.

MANITOBA

Lynn Lake—MORRISON, WILLIAM FRANCIS. (M). Resident geologist, Lynn Lake property, Sherritt Gordon Mines, Ltd.

ONTARIO

St. Thomas—CLAUS, SPENCER. (J). Geologist, Oliver Iron Mining Co.

Toronto — HEWITT, DONALD FRANK. (C/S—S-J). Geologist, industrial minerals, Ontario Dept. of Mines.

QUEBEC

Arvida—GILLIS, JOHN MAURICE. (M). Supervisor, caustic plant, Aluminum Co. of Canada.

CHIHUAHUA

Chihuahua — BROWN, SEVERN PARKER. (J). Assistant geologist, American Smelting and Refining Co. NESBITT, MARTIN BECERRA. (M). General superintendent, El Potosi Mining Co. ZANGWILL, AYRTON ISRAEL. (M). Assistant superintendent, Avalos smelter, American Smelting & Refining Co.

BRAZIL

Sao Paulo—SCHMIEGELOW, EGON. (C/S—S-J). Metallurgical engineer, Cia. Brasileira de Material Ferroviano.

CHILE

Chuquicamata — HENDERSON, HARRY BACKUS. (R-M). Test engineer, Chile Exploration Co.

GOLD COAST COLONY

Accra — NICHOLLS, WILLIAM. (R-AM). Mining engineer, Consolidated African Selection Trust, Ltd.

SIERRA LEONE

Freetown — MACKENZIE, GORDON BRIAN. (M). General manager, Sierra Leone Development Co.

INDIA

Bombay—MEHTA, R. K. (M). Technical director, Kamani Meta's & Alloys, Ltd. and Kamani Engineering Corp.

Kolar Gold Field—RAJAN, V. M. SUNDARA. (M). Chief assistant surveyor, Nundydroog Mines, Ltd.

QUEENSLAND

Mt. Isa—SHER, LESLIE JAMES. (M). Foreman, cottrell & baghouse, Mt. Isa Mines, Ltd.

VICTORIA

Melbourne—GLEN, MALCOLM. (M). Field engineer, North American Cyanamid, Ltd.

STUDENT ASSOCIATES

Elected May 18, 1949

Charles Orville Allen.....Univ. S. Calif.
Jerry Bob Allen.....Penn. State
Nicholas Dean Altimus.....Penn. State
Jerome Anton Andruske....Univ. Kans.
Raymond Richard Angstadt.Penn. State
Joseph S. Arnold, Jr...Univ. Pittsburgh
Harry Harper Bauer.....Penn. State
Rixford Allan Beals.....Univ. Calif.
Frank Patrick Bence.....Penn. State
Charles Richard Benson.....Ohio State
Alan M. Bieber.....Okla. A&M
Charles A. Bittmann..Bklyn. Poly. Inst.
Clarence W. Bowlby...Colo. Sch. Mines
Max Irving Brown.....La. State Univ.
Stuart S. Brown.....Melbourne Univ.
Robert Winslow Brumberg. Penn. State
Douglas F. Bulloch...Bklyn. Poly. Inst.
Jerry Dean Bullock.....La. State Univ.
Furman H. Burge, Jr..S. Dak. Sch. Mines
Robert Gustav Carlson..Bklyn. Poly. Inst.
Rodney Harvey Chow....Univ. S. Calif.
Allan Abbott Coburn..... Penn. State
Arnold Gaylord Connor.....Univ. Minn.
Charles James Conrad.....Penn. State
Allen L. Crockford.Camborne Sch. Mines
Coral Roland Dardenne..La. State Univ.
Laroy Dean, Jr. .....Univ. S. Calif.
Rafael K. De Bie.. Carnegie Inst. Tech.
Kramer Richard Duhe..La. State Univ.
Frank Earl Eckert, Jr.....Penn. State
Jack Walton Edwards.Texas Tech. Coll.
John Noble Ellis.....La. State Univ.
Courtland Harold Essex. Univ. Wichita
Charles Anthony Fabo.Univ. Pittsburgh
Alfred Norman Fanning (Rein.)
Mo. Mines & Met.
Chester Ernest Farrow, Jr...Okla. A&M
Robert Louis Folkman....Cornell Univ.
William D. Fugit.....Univ. Nev.
George C. Gabel.....Penn. State
Melih Izzet Genca.....Colo. Sch. Mines
Dan Kelly Gilbert..... La. State Univ.
Donald Flatau Goldsmith. Ill. Inst. Tech.
Samuel Goodman.....Wayne Univ.
Sol Gorenbein.....CCNY
Jack Edward Haflinger.....Penn Coll.
Jonathan H. Halsey (Rein.).Univ. Calif.
Robert L. Halvorsen.....Penn. State
Robert Kay Hansen.....Penn. State
Carl Dwayne Hanson.....Penn. State
Thomas Jakie Harp.... La. State Univ.
Francis David Hart.....Lafayette Coll.
John Edward Heim.....Univ. Wisc.
John Stewart Henkel..Melbourne Univ.
Robert A. Herstine.....Penn. State
Warren Henry Hinks, Jr.... Penn. State
Leonard M. Hogan....Melbourne Univ.
Perry Lanson Holsinger.... Ohio State
Robert Dillon Horst...Univ. Pittsburgh
Patrick Wendell Huber.....Penn. State
Walter Prentiss Jensen .La. State Univ.
Raymond Alvin Johnson....Univ. Utah
Ernest Allan Jonas..N. Mex. Sch. Mines
John Joseph Justen.....Univ. Tex.
Bob Kazarian.....Univ. S. Calif.
Dean David Kerr.....Univ. Kans.
John Stuart Koegle.....Ohio State
Arthur Andrew Krieger....Univ. Nev.
Ulf S. Landergren..Carnegie Inst. Tech.
Warren E. Latimer.....Penn. State
Robert William Lawrence...Penn. State
Eric J. Layton.....Melbourne Univ.
James Clark Leisk.....La. State Univ.
Robert Vinton D. Lord, Jr.... Iowa State
John Dabney Lung.....Univ. S. Calif.
Sidney Marcus.....CCNY
Roy C. Martin.....Ill. Inst. Tech.
Garrett Andrew Mayon.La. State Univ.
John Alfred McCabe.....Okla. A&M
James William McCarrick, Jr.
La. State Univ.
K. David McKay.....Melbourne Univ.
Charles Allan McLeish.....Univ. B. C.
Raymond N. McNally.Melbourne Univ.
Arthur Theodore Mengel....Penn. State
Robert J. Menzie.....Penn. State
Robert E. Merkle.....Penn. State
John Richard Miller.. Mo. Mines & Met.
George Montgomery Mitchell.Penn. State
Cecil Corter Miller.....St. Louis Univ.
Fred Edward Moore.....St. Louis Univ.
Lester Lazell Neblett, Jr..Univ. S. Calif.
Carl Godfrey Nelson.....Univ. S. Calif.
Roman Nowicki.....Wayne Univ.
Paul Francis Newton....Univ. S. Calif.
Dwight Conlan Oglesbee...Univ. Kans.
Albert Fred Peterson, Jr. La State Univ.
John Anthony Potenza.....Penn. State
Gerald Edward Rennick
Univ. Pittsburgh
John Peter Rokosh.....Univ. B. C.
Ronald George Rosenkranz.Univ. Wisc.
Robert Duane Saltsman Lafayette Coll.
David Albert Sanders.....Ohio State
William Dave Sauer.....Univ. S. Calif.
John Frederick Schnars....Penn. State
Richard Joseph Schneider
La. State Univ.
Edward Stewart Shea, Jr.
La. State Univ.
William Clarence Schoof.Univ. S. Calif.
Richard Charles Sennett....Penn. State
William Harvey Shaffer.Lafayette Coll.
Keith Darden Sheppard (Rein.)
Penn. State
Clark Merrill Sherwood.....Penn. State
Robert Harvey Shrut .Univ. Pittsburgh
William Reinarts Sittig....Penn. State
Charles Eugene Sonntag, Jr.
La. State Univ.
Henry Roy Speyrer.....La. State Univ.
Henry Lee Steadman.....Penn. State
Robert Thomas Thurston .....MIT
William R. Thurtle.....Ill. Inst. Tech.
Helmuth Timm.....Univ. Okla.
George Vacek, Jr.....Ill. Inst. Tech.
Jim Wilson Vidrine.....La. State Univ.
Roland D. Wardell.....Univ. Minn.
George Patterson West.....Penn. State
Charles Campbell White.. Univ. Otago
James Leland White.....Univ. Kans.
James Ronald Williams, Jr. Penn. State
Robert Arthur Winslow.....Penn. State
H. B. Wofford.....Univ. Kans.

American Institute of Mining and Metallurgical Engineers

No Directory will be published in 1949, so the following list of AIME officers and committees is being printed in the three journals. It includes the committees of the Mineral Industry Education and Mineral Economics Divisions. Committees of other Divisions appear in Section 1 of the Branch journals as follows:

Magazine	*Division*
MINING ENGINEERING	Coal Industrial Minerals Extractive Metallurgy Minerals Beneficiation Mining, Geology and Geophysics (Not yet appointed)
JOURNAL OF METALS	Institute of Metals Iron and Steel Extractive Metallurgy
JOURNAL OF PETROLEUM TECHNOLOGY	Petroleum

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1949

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VICE-PRESIDENT AND TREASURER

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Sinking with the Hydro-mucker at Mather "B" Shaft

By J. S. WESTWATER,* Junior Member, AIME

The Mather mine of The Negaunee Mine Co. embraces nearly all of Sections 1 and 2, T47N, R27W, within the limits of the cities of Negaunee and Ishpeming on the Marquette iron range of Michigan's Upper Peninsula. Operations of the Negaunee Mine Co., the stock of which is jointly owned by Bethlehem Steel Co. and The Cleveland-Cliffs Iron Co., are under the direction of the latter company.

The opening of the Mather mine was described by C. W. Allen.†

Mather "A" shaft was completed to an initial depth of 2352 ft in early 1943. The shaft capacity of over one million tons yearly is in prospect of being realized by orderly development and mining, and sinking of the second or "B" shaft then started in accordance with previous planning. Mather "B," also a vertical shaft, was designed as a twin to Mather "A" shaft and the sinking technique developed at Mather "A" was utilized in "B" shaft operation.

Shaft Sinking General Plan

An average depth of 16 ft of sand overburden at the shaft site was cleared away by bulldozer, and the shaft approximately 15 by 20 ft in rock opening put down to an initial depth of 87 ft. Mucking of the broken rock to this depth was accomplished by an Insley excavator equipped with a ½ yard capacity Owens clamshell bucket. Addition of 200 lb weight to each lip of the clam bucket provided adequate "bite" to dig the broken diorite and jasper rock. Temporary heavy wood bearers were placed across the shaft mouth to support the steel shaft sets which were hung in this initial 87 ft depth, aligned accurately and concreted in place from the lowest set upwards. The permanent steel headframe foundations and three service tunnel openings to the shaft were also concreted in the upper section to the shaft collar.

A temporary wood headframe was then erected over the shaft mouth, and initial hoisting machinery assembled consisting of a Lake Shore 12 ft diam single drum grooved for a 1⅜ in. rope and powered by two 400 hp motors to attain hoisting speed of 1100 fpm. Compressed air was obtained from a Nordberg 2 stage tandem horizontal type with a capacity of 1500 cfm. An enclosed landing platform at the shaft collar was equipped with a rotary dump and pocket for discharge of the rock into a Euclid 15 ton truck.

On completing assembly of this equipment and installation of a special cage in the shaft, sinking was resumed on a three shift, six days per week

San Francisco Meeting, February 1949.

TP 2596 A. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before July 30, 1949. Manuscript received Jan. 26, 1949.

* Superintendent, Mather Mine["B" Shaft, Cleveland-Cliffs Iron Co., Negaunee, Mich.

† C. W. Allen and L. C. Moore: Opening the Mather Mine. *Trans.* AIME (1945) 163, 293.

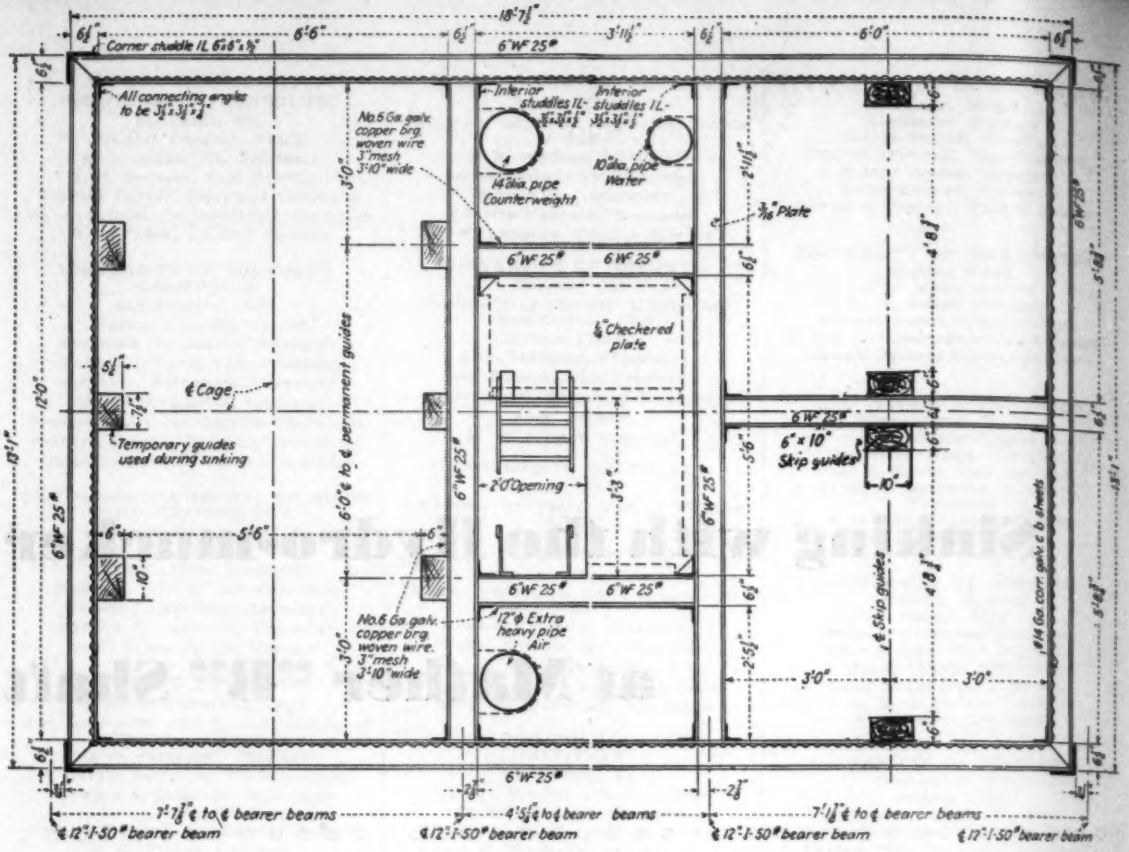
Corner studdle IL 6'6"x½"
6"WF 25#
Interior studdles IL 3½"x3½"x½"
All connecting angles to be 3½"x3½"x½"
No.6 Ga. galv. copper brg. woven wire 3" mesh 3'10" wide
14"dia. pipe Counterweight
10"dia. pipe Water
⅜" Plate
¼" Checkered plate
℄ Cage
Temporary guides used during sinking
6"x10" Skip guides
2'0" Opening
12"φ Extra heavy pipe Air
No.6 Ga. galv. copper brg. woven wire. 3" mesh 3'10" wide
#14 Ga. corr. galv. c.b. sheets
℄ Skip guides
7'-7⅞" ℄ to ℄ bearer beams
4'-5¾" ℄ to ℄ bearer beams
℄ 12"-I-50# bearer beam

FIG 1—Steel shaft sets.

schedule. The equipment used and sinking procedure established in the Mather "A" project was fully covered by C. W. Allen in his earlier paper. Mather "B" shaft, while re-using the same or employing similar equipment and following generally the pattern adopted for "A" shaft, has made use of the advantages offered by the tungsten carbide tipped drill bit and of a hydraulically operated clamshell bucket in mucking operations.

Fig 1 shows the arrangement of steel shaft sets which are installed on 7 ft centers. The sets are hung as shown from 6 by 6 by ½ in. angle studdles and are blocked by a reinforced concrete ring, bevelled below the set to the rock walls, or by sacked concrete jammed between the beams and the rock. Normally every other set is placed with a bevel concrete pour and the alternate sets blocked with the concrete filled sacks, however in sections of the shaft possessing unstable wall conditions, each set is installed with the bevel pour.

Fig 2 illustrates sets that are of Bethlehem Mayari-R steel, 6 in. 25 lb *H*-beams, and are credited with a corrosion resistance 6 times that of ordinary steel. The arrangement provides for a 4 compartment shaft consisting of a cage compartment 6 ft 6 in. by 12 ft 0 in. inside, a ladder and pipe compartment 3 ft 11½ in. inside and 12 ft long, and 2 skip compartments 6 ft 0 in. by 5 ft 8¾ in. The cage and skip compartments are separated by the ladder compartment. In addition the skip roads are completely sealed from the other two compartments by a partition consisting of ¾16 in. steel plate which extends the width of the shaft and is made air tight to the shaft walls by the compaction of concrete filled sacks between the edge of the plate and the rock. In normal ventilation the cage road is downcast with fresh air and the skip compartments upcast carrying the discharge air to prevent freezing of the skip dumps. In case of fire underground no contamination of the fresh air entry is possible with the plate seal. Corrugated lath, No. 14 gauge, copper-bearing galvanized sheets, fully enclose the shaft perimeter. The sheets are made fast to the 6 in. *H*-beams set by bolting with a specially designed clip.

In the ladder compartment, wire mesh of No. 6 gauge, 3 in. square openings, is fastened to the double cage guides, extending wholly across the opening of the ladder road proper. At the sollar sets of the ladder road, 2½ in. guard angles are placed across the interior studdles to prevent mis-step.

Two pairs of wood guides 6 by 10 in. will be installed in the cage compartment to serve the permanent aluminum double decked cage, replacing the 5½ by 7½ in. single guides now serving the sinking cage. The same dimension wood runners will serve the skips which will be equipped with rubber tired guide wheels to minimize wear of the supplementary steel guide shoes.

The ladder compartment is divided into three parts, the center serving as the ladder road proper, and each end

compartment for pipe and power cable installation. A 14 in. cage counterweight pipe is being installed as sinking progresses and serves as ventilation conduit from a size 45 Sturtevant planovane fan which is capable of producing 3000 cu ft of air per minute through 2000 ft of 14 in. pipe against a maximum static pressure of 20 in. of water. An adjustable by-pass arrangement is provided between the fan and the shaft conduit to permit blowing or exhausting action of the fan which rotates in one direction only. A 10 in. diam water discharge line and a 12 in. diam compressed air line are to be installed, and the weights of these large diameter pipes are carried only by the bearer sets installed at intervals of 150 ft. These bearers are 12 in. 50 lb *I*-beams and 4 beams constitute a set. They are placed across the short dimension of the shaft, two immediately adjacent to the end pieces of the shaft sets and two over the interior dividers. These bearers are of sufficient length to jut into niches cut in the shaft walls. The *I*-beams are bolted to the shaft set and the combination is then embedded in a solid ring of concrete.

A sinking cage, shown in Fig 3, has a platform 5 ft 6 in. by 9 ft 0 in. and is equipped with safety catches and a steel crosshead which permits lowering to a distance of 30 ft below the bottom shaft set. The use of this type cage permits movement of men and supplies with comparative safety and in the mucking operation, hoists a 2¼ cubic yard capacity car. In conjunction with the cage is a 2 yard capacity mucking tray (Fig 4), equipped with a bail and possessing two hooks at the front, which is hoisted by an Ingersoll-Rand K4U air hoist having a rated capacity of 3500 lb rope pull at 80 lb pressure and a lift speed of 95 fpm. The cage is lowered to the muck pile and the hooks of the tray engage two dumping bars on the cage permitting the tray to discharge its load through an opening in the side of the cage into the car blocked in place on the cage platform.

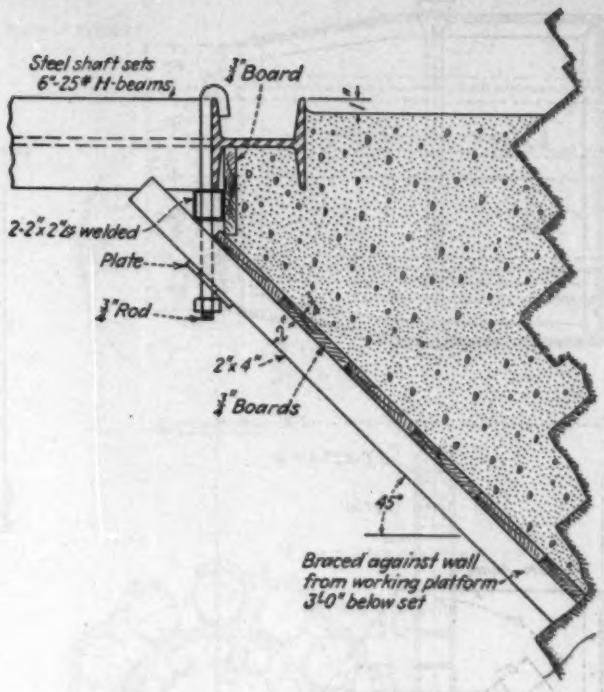

FIG 2—Method of concreting steel shaft sets.

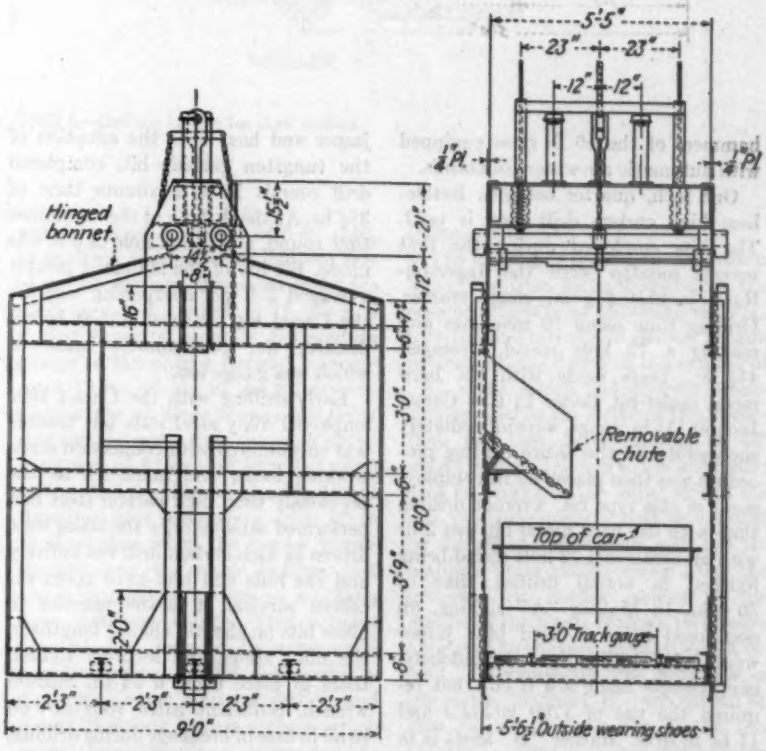

FIG 3—Sinking cage.

Details of Sinking Operation

DRILLING

Iron formation or jasper was penetrated from near the collar to the 1650 ft depth. A 7 ft depth of cut was taken consisting of 72 holes drilled breaking to a *V*-cut. The drill machines used are the Ingersoll-Rand J-50 Jack-

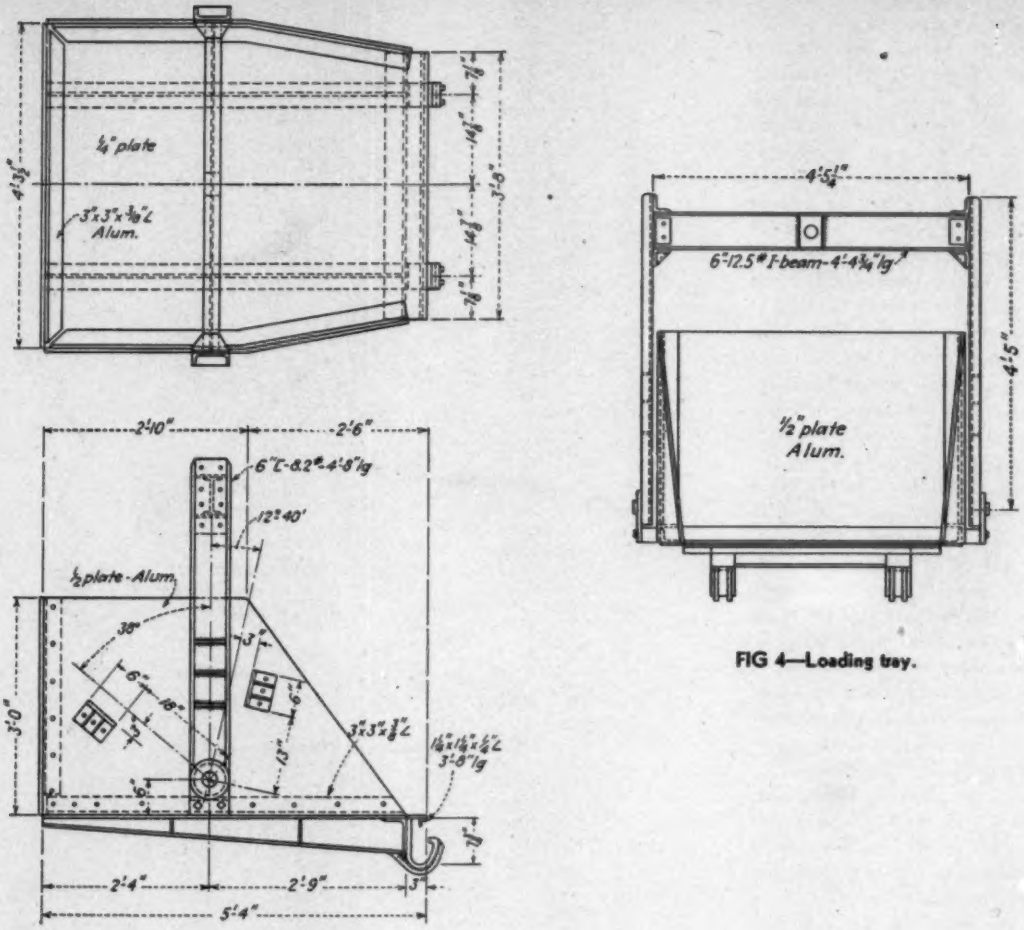

FIG 4—Loading tray.

hammers of the 60 lb class equipped with automatic air-water backheads.

One inch, quarter octagon, Bethlehem high carbon drill steel is used. The bits employed during the first several months were the Ingersoll-Rand jackbit 2¼ in. gauge starter. Drilling time using 10 machines and making a 72 hole round, averaged 4¼ hr. Tests made with the hard metal insert bit, Series 13 CS4 Carset Jackbit 1½ in. gauge, were immediately successful and the entire drilling procedure was then placed on full employment of this type bit. Average drilling time with the hard metal bit was 2 hr with an occasional 72 hole round being finished in actual drilling time of 50 min. In Mather "A" sinking, an occasional layer of hard blue jasper would boost drilling time tremendously, one example being a 4 ft cut that required the use of 1700 jackbits and 11 hr to drill. Mather "B" shaft is in the same geological horizon and has penetrated similar hard layers of blue jasper and has, since the adoption of the tungsten carbide bit, completed drill rounds in a maximum time of 3½ hr. At the drilling of the maximum time round, a depth of hole of 8 ft was made. Bit life of the standard jackbit averaged 2 ft per sharpening whereas the Carset bit produced 135 ft before discard, the predominant cause of which was gauge loss.

Early drilling with the Carset bits, employed alloy steel rods but trouble was encountered with connection studs working loose, and inasmuch as the previously used high carbon steel had performed satisfactorily the studs were driven in high carbon drill rod entirely and the rods and bits have given excellent service. It is the practice to place bits on the rod change lengths in the mine shops and securely tighten them in place using a 24 in. Stillson wrench. Sufficient extra rods are on hand in case of breakage during drilling, eliminating any necessity for bit changing by the miners during the round. The drill machines, rods, and bits are thoroughly inspected after every drilling period. In the case of bits, a gauge test is made and if found to have lost to the extent of 1/16 in. the bit is removed. In the case of longer lengths, a bit is interchanged from a short length and a new bit placed on the short lengths. This ensures greater gauge on the shorter rods and has eliminated to a great extent loss of time due to stuck steel.

Bit discard is due preponderantly to gauge loss which is quite severe in down holes drilled in the extremely abrasive, high silica jasper formation. It has been found, however, that the bits removed from service because of gauge loss produce a footage equal to that given in the shaft when used later in drilling horizontal or upper holes at other company mines.

Air was supplied to the machines in early drilling by means of a header equipped with 12 I-R line oilers of 1 pint capacity, and necessary air and

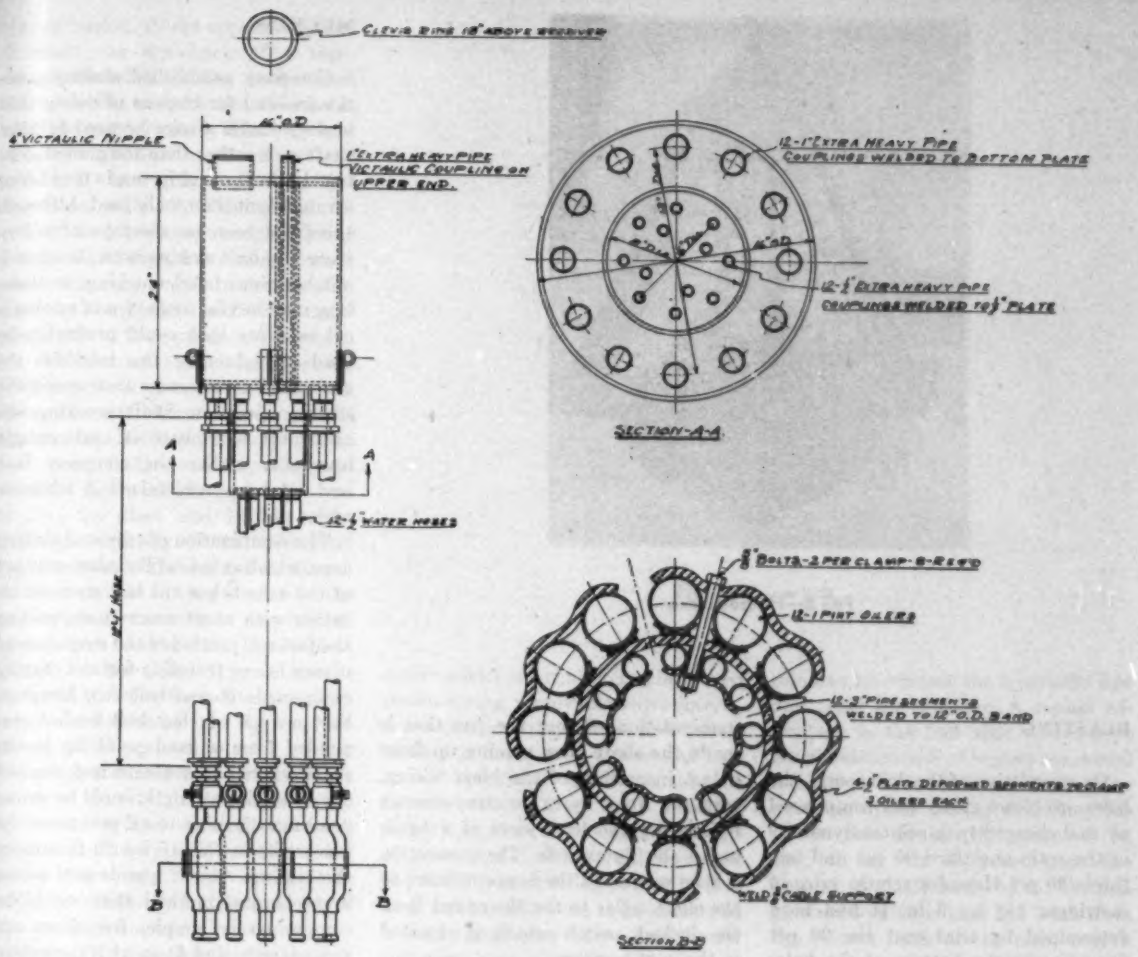
CLEVIS RING 18" ABOVE RECEIVER
4" VICTAULIC NIPPLE
46" O.D.
1" EXTRA HEAVY PIPE VICTAULIC COUPLING ON UPPER END.
24"
12-1" EXTRA HEAVY PIPE COUPLINGS WELDED TO BOTTOM PLATE
12-½" EXTRA HEAVY PIPE COUPLINGS WELDED TO ⅜" PLATE
SECTION A-A
12-½" WATER HOSES
84" HOSE
⅝" BOLTS-2 PER CLAMP-6 REQ'D
12-1 PINT OILERS
12-2" PIPE SEGMENTS WELDED TO 12" O.D. BAND
4-¼" PLATE DRILLED SEGMENTS TO CLAMP 3 OILERS EACH
WELD ⅜" CABLE SUPPORT
SECTION B-B

FIG 5—Drilling header for shaft sinking.

water connections. The header was fed by a 6 in. line and a connecting 3 in. rubber hose. A K4U utility hoist mounted on a shaft set raised and lowered the header as desired. From the header, air was supplied to the drill machines by 30 ft lengths of 1 in. hose to ensure adequate volume and pressure. It was found, however, that insufficient lubrication was being given the drill machines, and that the 1 in. hose was awkward and cumbersome in handling the machines, therefore, a header or manifold of different design was built at the mine shop. This header as shown in Fig 5 consists of an air receiver, 16 in. diam pipe 24 in. long, through which passes, completely enclosed, a 1 in. extra heavy pipe for water. Twelve 1 in. air hoses are connected to the bottom of the air receiver near the outer circumference. Likewise twelve ½ in. water hoses lead from the water receiver a section of 8 in. pipe, 12 in. long, welded to the bottom side of the air receiver. These air hoses are 12 ft long and connect to 12 oilers of 1 pint capacity securely fastened in groups of three, in a circular arrangement 19 in. od. As in the upper connection the air lines occupy the outer ring and the water lines the inner circle. Air hoses of ¾ in. lead from the oilers to the drill machines. The lengths of these machine hoses vary from 12 to 17 ft so that when the header is suspended over the shaft bottom just above head height, the corner machines have sufficient length. This octopus type header, designed by mine engineer Harry Swanson, has delivered adequate air, water, and lubrication and has as its strong feature a shape which provides no difficulty in traversing the constricted pipe compartment of the shaft when being raised out of the way of the blast.

The number of holes drilled per blast has varied from 65 to 85 matching the ease or difficulty of obtaining satisfactory fragmentation and cleanly broken bottom. It was found in the sinking of "A" shaft, which had its long dimension parallel to the strike of the formation, that the slates dipping at 45° would spall and slough to a semicircular arch especially on the upper side of the dip. Not infrequently that side would be drilled at the corners and the remaining holes pulled away from the proposed rib outline almost to the middle of the shaft. Despite this buffer approach the severe arching would repeat. To counteract this weakness in the slates, Mather "B" was turned so that the short dimension of the rectangular outline was presented to the strike and greatly reduced overbreak has been experienced.

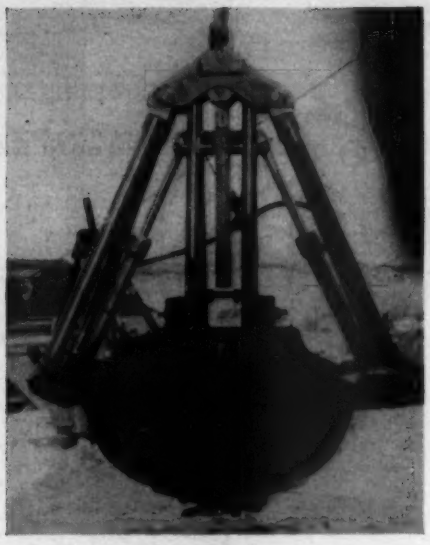

FIG 6—Hydro-mucker.

BLASTING

On completion of the drill round, the holes are blown clean with compressed air and charged with gelatin dynamite in the ratio one third 90 pct and two thirds 80 pct Hercules gelatin extra in cartridges 1¼ by 8 in. It had been determined by trial that the 90 pct dynamite in the bottom of the holes produced a cleaner blast bottom in the dipping layered jasper formation. The primer cartridge is loaded as the second stick in the hole, and contains a delay electric blasting cap. These delays range from the three available short period or millisecond delay to the standard 12th delay. Lead wires from the individual blasting caps are bunched in groups of not over 5 by color and are twisted in a tight connection to one of two bus wires suspended by wooden pegs directly over the blast area. These bus wires are tinned, bare copper No. 14 gauge wire. A No. 14 duplex wire leads above the blast area to connect to a No. 4 rubber covered Tyrex cable which extends the length of the shaft to a blasting station in a concreted service tunnel near the shaft collar. The blast is fired from a locked safety switch, the key of which is in the possession of the shaft boss. Alternating current is used to fire the round, 110 v was used for the first 1600 ft and 220 for the succeeding lift. The junction of the No. 14 Duplex to the No. 4 is made on a box equipped with wing nuts for easy manipulation. Before the junction is made the shaft boss, coming up from a last inspection of the blast wiring, tests the No. 4 Tyrex for stray current by placing the lead wires of a squib across the Tyrex ends. The connection is then made and the boss continues to the shaft collar to fire the round from the locked switch which is shunted in the open position.

It was found in the initial drilling with the hard metal bits that because of the uniform small diameters of the holes, 1½ in. as compared to the standard jackbit inverted tapered hole ranging from 2¼ in. at the collar to approximately 1¾ in. at the bottom, much less dynamite could be loaded in the 1½ in. hole. The ratio was approximately 50 pct. On blasting however, equal if not better fragmentation was obtained from the smaller powder loads. One explanation offered for this seemingly unbelievable result is that perhaps because of the uniformly parallel sided hole of the hard metal bit, the gas pressure is retained slightly longer than in the outward sloping conical hole produced by the larger jackbits.

In the hand mucking stage of sinking, good fragmentation was obtained in the jasper with a 7 ft cut. Use of the hydro-mucker eliminated the necessity for fine fragmentation and the cuts were deepened to 9 ft. Wall stability is now the governing factor in depth of round blasted.

MUCKING

Company established sinking practice insisted for reasons of safety that in deep shafts a cage be used to hoist shaft rock rather than the guided sinking bucket or skip and trap door arrangement commonly used. Although there has been no shortage of volunteers for shaft sinking with its attendant laborious hand mucking, we have long searched for some type of mechanical mucking that could profitably be used in lightening the load for the crews and at the same time speed the sinking operation. Shaft mucking was about the last phase of underground operation within the company that had not been mechanized in whole or major part.

The combination of tray and sinking cage, which at times filled almost 40 pct of the area below the last steel set together with unset concrete supporting the last set, precluded the employment of any heavy traveling form of digging equipment. It was believed, however, that a type of clamshell bucket suspended from a fixed point up in the shaft where the concrete had reached a good setting strength, could be swung pendulum fashion to all points on the muck pile in this 15 by 20 ft area at the bottom. Also it was desired to use a mechanical method that would be comparatively simple, free from appurtenances, and fit in with the equipment of the hand mucking tray system. If, as mechanical innovations are prone to do, failure of some part should occur, especially in the early stages, the shaft mucking could proceed in the previously accustomed fashion with little interruption.

In late 1947 trials were made with several types of industrial rope-reeved clamshell buckets actuated by various types of air and electric powered hoists. These experiments were made in a full scale model of 4 shaft sets on surface to simulate actual space conditions in the shaft and to work out any difficulties that could be foreseen before placement of the apparatus in actual service. Mechanically, the result of these trials was successful, but it was obvious that some training of the miners as clamshell operators would be necessary for successful combination of the placing and digging action of the equipment. At that time, however, acquaintance was made with the Milwaukee Hydrocrane Co., who utilized a bucket operated hydraulically in conjunction with their mobile crane. This

type of bucket, it was apparent, would eliminate the drawback of a rope-reeved clamshell pulling away from the muck pile, and, further, the bucket would be comparatively light in weight for easy movement by the miners and yet obtain a load without the restrictive directional pull of the digging line.

With the Hydrocrane people offering fullest cooperation, a layout was made which was very simple in design, adding only the bucket and a hydraulic power unit to the equipment already in the shaft for hand mucking purposes. The bucket as manufactured by the Milwaukee Hydrocrane Co., now the Hydrocrane Division of Bucyrus-Erie Co., was of ¾ yard capacity and operated by a single hydraulic ram of 2 5/16 in. diam and 19 in. stroke mounted vertically within the bucket yoke or frame. The bucket in dimension and outline is very similar to the conventional industrial rope-reeved bucket and weighs 1350 lb complete (Fig 6). A low head type of bucket equipped with two horizontal rams was being developed by the Hydrocrane people which seemed to offer greater advantage for shaft work with its lower center of gravity, but to initiate the hydro-mucker and work out the weak spots bound to appear, it was decided to use the vertical type rather than wait for satisfactory development of the low head bucket.

Hydraulic power for the bucket was provided by assembly at our General Shops of a unit consisting of a 20 gal oil reservoir tank, a small filter, a 15 hp motor, two hydraulic pumps and a control bank, all mounted within a base and bail to permit lowering through a pipe compartment 2 ft 5½ in. by 3 ft 11½ in. Fig 7 shows the original power unit built. A 15 hp, 220 to 440 v, 3 phase, 1800 rpm, drip proof motor mounted on a base plate drives two Blackhawk P-197 pumps by *V*-belt and pulleys. These 3 gpm pumps, mounted on brackets for quick belt adjustment or removal, are 6 cylinder radial style piston type pumps, wherein radial motion is transferred into reciprocating motion by means of a wobble plate. The pump is capable of pressures up to 6000 psi. Good bucket digging action and pump life have been obtained from operation of the pumps at 5000 lb pressure and a speed of 1400 rpm. Bearings of the pump are antifriction type, throughout. At present an improved type of pump classified as P-237 is being used. This latest design has eliminated several causes of failure in the earlier model and contains an improved piston spring, removable valve inserts, Amco metal rocker caps and swivel sockets on ends of the pistons and uses no soft packings. At the outset, the rigorous demands of shaft mucking which comprised practically a straight 17 hr usage out of a total 27 hr cycle, the P-197 pumps failed in four to five days service. This was to be compared to an average year and a half life for the same type pump used in the mobile crane. This feature was extremely critical and at first jeopardized an otherwise highly successful operation. Thorough and constant observation by mine mechanic, John Nigra, singled out the prime causes of failure and with splendid cooperation from the Blackhawk Manufacturing Co., these weaknesses were rectified and since the two pioneer models of the P-237 were introduced into the shaft for proving ground purposes, no further trouble has been encountered.

FIG 7—Power unit.

The procedure followed in removing shaft rock with the hydro-mucker is essentially the same as that of hand mucking, that is, the 2 yard tray is filled by the clamshell bucket and the contents of the tray dumped through the side of the cage into the 2¼ yard capacity car. A major advantage of the hydro-mucker which should be emphasized is the extremely short preparation time necessary for the start of the mucking cycle. Since the tray and bucket are the only two pieces of equipment removed during other phases of the sinking operation, it is a matter of minutes to connect the hydraulic line and begin the mucking. A second advantage is the fact that at no time have maintenance or repairs consumed more than a few minutes during the mucking cycle. During mucking, the power unit is usually mounted on the 4th or 5th shaft set from the bottom. A 65 ft long, ⅜ in. id, high pressure hydraulic hose line leads from the control bank of the unit, over a stationary reel from which loops are paid off as additional length is necessary, down to the bucket. A miner sitting on the lowest shaft set, which is customarily 20 to 30 ft above the muck pile, opens and closes the bucket by manipulating two draw cords attached to a short crossbar or yoke on the control bank lever in the power unit above. He has a commanding view of the operation and is up out of the way. The bucket is raised or lowered by a reversible air hoist of the K4U type which is operated by a miner standing on the muck pile holding two draw cords attached to a yoke on the hoist control valve lever. A ⅝ in. wire rope which raises the bucket, is double reeved by being passed through a 10 in. traveling block above the bucket, and is then dead-ended on the shaft set above that on which the air hoist is mounted.

When the tray is full, the bucket is swung to one side and permitted to dig itself into the broken rock to maintain an upright steady position. A removable pin is quickly slipped out of the clevis on the bucket yoke, and the rope

Table 1 . . . Monthly Progress Record

Month	Days Worked	Footage		Cuts Blasted	Steel Sets	Sets Concreted	Cubic Yards Concreted	Cubic Yards Rock Hoisted	Cubic Yards per Foot	Advance per Day
		Advance	Depth							
Prior 1947.....		128	128	19	15	12	494	640		
November 1947.	26	88	216	16	13	7	33	1,446	16.4	3.4
December 1947..	26	107	323	20	14	7	39	1,466	13.7	4.1
January 1948...	26	142	465	26	19	9	41	2,210	15.5	5.5
February 1948..	25	101	566	21	16	10	66	1,606	16.1	4.0
March 1948.....	27	118	683	23	17	17	125	1,900	16.1	4.4
April 1948.....	26	144	827	27	20	12	122	2,409	16.7	5.5
May 1948.....	25	138	965	26	20	13	44	2,480	18.0	5.5
June 1948.....	26	138	1,103	28	18	15	87	2,311	16.7	5.3
July 1948.....	26	140	1,243	28	21	12	82	2,680	19.1	5.4
August 1948....	20	112	1,355	17	16	9	68	2,238	20	5.6
September 1948.	24	153	1,508	19	22	11	105	2,810	18.4	6.3
October 1948....	26	170	1,678	24	24	13	101	3,198	18.8	6.5
November 1948.	21	138	1,816	17	19	10	95	3,054	22.1	6.6
December 1948..	25	145	1,961	19	21	13	105	3,354	20.3	5.8

attached to the bail of the tray by the same pin and a similar clevis. The cage is lowered to the dirt pile and the tray raised. The position of the dead end of the tray hoisting rope, on the divider separating the cage and ladder road compartments, swings the tray to the cage allowing hooks on the tray to slide up along the side of the cage ensuring engagement on the dumping bars. After dumping its load, the tray is swung to a favorable position on the pile, the pin and block removed, attached to the bucket and the loading operation repeated. Three to four men swing the bucket into digging position and in the 15 by 20 ft area very little picking down is necessary since the straight sides and lip of the bucket fit the square corners of the shaft.

A 9 ft cut in the slates produces an average of 92 trays of 2 yards each or 185 yards of broken rock. All but ½ to 1 tray at the final clean-up is handled by the hydraulic bucket. It is advantageous to clean the bottom thoroughly before starting to drill, thus eliminating to a great extent hole-starting and stuck-steel difficulties caused by loose rubbly bottom. It was customary to pick and moil 2 to 3 cars to clean the bottom in the hand mucking stage, but the positive action of the 4 ton closing pressure of the clamshell teeth acts as a 5 pointed moil and seldom is it necessary to hand shovel more than ½ tray. Large slabs or blocks pried from the walls in trimming following a blast are easily handled by the bucket.

The original vertical bucket required 2 to 4 sec to close on a load and 13 sec to open. Trials with the double ram, low head, modified, materials-handling bucket duplicated the closing time and had a 6 sec opening time but this type bucket did not possess as good digging qualities in the rock as the upright bucket. Therefore, at the mine shops the two rams each 1¾ in. diam, 13 in. stroke were mounted inclined at 45° on the vertical bucket replacing the larger 2⅜₁₆ in. diam, single vertical ram to produce a hybrid bucket possessing good digging action and quick opening.

Performance data shows hand mucking production with 10 men as 8 yards per hour up to the time the hydro-mucker was introduced, which was August 1948. Since that time the yardage per hour has risen steadily as improvements were made and the system became trouble free and the month of December 1948 showed an output of 11 yards per hour which is an improvement in excess of 35 pct over hand mucking. A 10 to 11 man crew was necessary in hand mucking and while a 7 man crew is adequate with the clamshell, a 9 man crew is regarded as most efficient for all phases including drilling, shaft steel placement, and concreting work.

A monthly progress record, Table 1, gives information from which the general story of the sinking operation to date may be drawn. For example, January of 1948 had very good wall conditions requiring solid concreting of every other set according to plan and an advance of 142 ft was made. March had the same type of jasper rock from a hardness standpoint in drilling, but the blocky characteristic was much more pronounced, which factor demanded concreting of practically every set and required placement of 125 yards of concrete as compared to 41 yards in January. The cubic yards of rock hoisted per foot of advance was consequently high in March and the resultant gain was 118 ft as contrasted to the straight going 142 ft of January.

The comparison summary of shaft sinking figures between Mather "A" and Mather "B" shafts, which were put down through similar rock conditions and in which "B" used the same methods and equipment at the beginning, shows a marked increase in footage advance per month. The first year of sinking Mather "A" produced an average monthly advance of 99 ft, and during the second year an average of 107 ft was made with a monthly high of 124 ft of completed steel and concrete shaft. In the first year of Mather "B" sinking an average of 137 ft per month has been made with a high month of 170 ft on the three shift, six day per week schedule. It is further interesting to note that the average monthly advance of Mather "B" prior to the adoption of the hydro-mucker was 129 ft and in the past four months of clamshell usage the average was 151 ft of shaft advance. Since the sinking crews were of comparable caliber and of slightly less number in "B" shaft sinking and rock conditions have been similar to those of Mather "A," the helpful factors have been the hard metal tipped Carset bits and introduction of the hydraulic clamshell bucket for mucking.

The speed up and ease of mucking combined with the simplicity and low cost of the so-called hydro-mucker are important factors recommending expanding use. There seems to be little question that employment of this equipment should produce similar worth-while results in other sinking jobs no matter what their size or shape, the one evident requirement being that the shaft shall be vertical or nearly so.

Diamond Drilling Quartz-feldspar Intergrowths

By L. C. ARMSTRONG,* Member AIME

Twice in the past two years and in two widely separated localities—one near Williamsville, Mo., and the other in the Allard Lake district of Quebec—the Contract Drilling Division of the Longyear Company has experienced perplexing difficulties in drilling granitoid igneous rocks. The rocks in both places are characterized by appreciable amounts of feldspar intimately intergrown with quartz particles of microscopic dimensions. The disappointingly slow rate of advance and the exceptionally high bortz loss, obtaining during the life of these two projects, are ascribed largely to the resistance to abrasion of these two minerals when arranged in this intergrown pattern. Whether all intergrowth-bearing igneous rocks would offer extraordinary resistance to bit progress cannot be stated with assurance on the strength of these experiences alone. More data on diamond drilling this sort of material in a number of localities are needed before any generalizations can be made safely.

It should be pointed out that the lack of time and funds made it impossible for the writer, or other persons interested in correlating the details of bit histories with the kinds of rocks drilled, to be present during the performance of the contracts. However, the records kept in the normal course of drilling are deemed complete enough to be reportworthy. This information is presented here together with the findings made in studies of the rocks involved, with the thought that it may attract the comments of others who may have encountered similar refractory drilling conditions. A pooling of knowledge on the problem would be of material aid in devising more efficient cutting tools and procedures for penetrating this kind of rock.

Drilling Blastholes in the Williamsville Granite

Due to the unsatisfactory performance of pneumatic percussion drills in boring blastholes on a construction contract near Williamsville, Mo., tests were made with a diamond drill. The poor progress achieved and the excessive loss of diamonds, however, soon led to the abandonment of these trials. The diamonds, in all of the bits tried, lost their cutting edges and acquired a polish after being in service for distances ranging from a few inches to a few feet. Coring bits containing surface-set, whole-stone bortz in a tungsten powder matrix were employed. The cost assignable to bortz wear averaged $7.17 per foot. This item of expense is several times the amount normally expended in diamond drilling in firm, unfractured, granitic rocks. Diamond-impregnated, tungsten carbide bits also were used without success. The diamond wear was at the rate of $0.98 per foot, but the diamond chips became too polished for further duty in less than a foot of drilling.

The Williamsville granite is pink and medium-grained. In the hand specimen, it has a fresh, sound appearance; it is seemingly composed almost entirely of interlocking grains of feldspar and quartz and has no discernible features suggesting a greater durability than is usually found in rocks with this composition and texture. Microscopic examination, however, reveals a predominance of feldspar grains with more or less regularly distributed and sharply angular masses of quartz (see Fig 1). The pattern assumed by the two minerals in this mode of occurrence is referred to as a "graphic intergrowth," as it bears some resemblance to the cuneiform characters of ancient Persian writing. The other principal constituents are quartz-free feldspars (microcline and orthoclase) and quartz. All of these constituents are in nearly equidimensional grains, ranging up to about 3 mm in length. There are also

FIG 1—Micrograph of granite from Williamsville, Mo. showing graphic intergrowths. White areas are largely quartz; those with black and grayish hues are mostly feldspar (orthoclase and microcline). Crossed nicols. Approximate magnification: X 40. (Reduced approximately one fifth in reproduction.)

FIG 2—Micrograph of anorthosite from Allard Lake district, Quebec. Plumose areas with vermicular banding are myrmekite. Grains with straight parallel lines are labradorite. Crossed nicols. Approximate magnification: X 40. (Reduced approximately one fifth in reproduction.)

San Francisco Meeting, February 1949.

TP 2580 A. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before July 30, 1949. Manuscript received Dec. 1, 1948.

* E. J. Longyear Co., Minneapolis, Minn.

Table 1 . . . Composition of Granite from Williamsville, Mo., and of Anorthosite from Allard Lake District, Quebec

	Williamsville Granite (Estimated, Per Cent)	Allard Lake Anorthosite (Estimated, Per Cent)
Quartz-feldspar intergrowth	(graphic) 61.0	(myrmekite) 22.0
Feldspar	(potash) 21.0	(labradorite) 75.0
Quartz	15.5	
Total quartz and feldspar	97.5	97.0
Magnetite	1.0	0.9
Green biotite	0.5	
Green hornblende		0.2
Muscovite	0.3	
Augite		0.2
Chlorite	0.3	0.2
Epidote		0.2
Ilmenite and/or magnetite		0.2
Secondary minerals (probably largely epidote, chlorite, and calcite)		1.0

many finer-grained, less abundant accessory minerals and alteration products, including magnetite, green biotite, muscovite, chlorite, and leucoxene. The amount of each mineral present, estimated in a study of two thin sections, is listed in Table 1, and to facilitate comparisons, the components of the Allard Lake anorthosite are also given.

Drilling in the Allard Lake Anorthosite

According to accounts received from others,* prospecting in the Allard Lake district of Quebec for ilmenite entailed drilling in anorthosite and ilmenite ore, and in gradations between these two classes of material. Where the rock was dominantly ilmenite, the cost per foot and the drilling speed were about what would be expected in drilling in the Canadian Shield. Sections of anorthosite, on the other hand, were reflected by a slower rate of penetration and by an increase in the destruction of bortz. In anorthosite, a new bit would cut satisfactorily during the first few feet of drilling, but almost without fail the diamonds would acquire such a smooth polish in a 10-ft run as to make the bit unfit for further use in this kind of rock. Frequently, however, it was feasible and economically worth-while to do additional drilling in ilmenite ore with these worn bits before returning them for resetting.

Whole-stone, surface-set bits with a beryllium-copper matrix were supplied for the work. All were EXT, a Canadian bit size, which, except for a slightly larger inside diameter, has dimensions identical with the EX size used in the United States. No attempt was made while conducting the work to maintain separate data on the results obtained in anorthosite and in ore, but subsequently, those associated with the project have made fairly accurate estimates which indicate a cost of $2.00 a foot, or more, as being chargeable to bortz loss while coring anorthosite. This figure is considered very large in relation to bit performances in drilling the average pre-Cambrian rocks in Canada.

As represented by pieces of drill core, the anorthosite is a pinkish gray rock with a large percentage of coarse, plagioclase feldspar grains attaining lengths of approximately 1 cm. Some smaller, irregular-shaped, hard, gray masses occur scattered throughout the rock, but as they are not conspicuous, they would probably be overlooked in a casual inspection. Under the microscope, it may be observed that three-fourths of the rock is labradorite and that the hard, gray masses, occupying much of the remaining one-fourth, are composed of myrmekite, a vermicular intergrowth of quartz and feldspar (see Fig 2). The areas of myrmekite consist of thin "wormy" layers of quartz alternating with similar layers of feldspar. As this intergrowth replaces the labradorite, it was probably formed after the crystallization of the labradorite but before end-stage, deuteric action had ceased. Inconsequential amounts of other minerals, such as magnetite, augite, hornblende, epidote, chlorite, apatite, and calcite are also present. Some of the epidote and calcite, and possibly one or two other minerals, occur in thin reaction rims around grains of labradorite.

Conclusions

There is little doubt that drilling the Williamsville granite and the Allard Lake anorthosite was affected adversely by the presence of quartz-feldspar intergrowths. On the basis of these two case histories, it is thought likely that rocks of this kind become progressively more unyielding as the content of quartz-feldspar intergrowth increases.

Feldspar is softer and more cleavable than quartz, and therefore would probably yield more readily to the shearing forces set up, when bortz are forced onto it and the bit rotated. For this reason, it is thought probable that a hole being drilled in rocks like those described above would be, at all times, a small fraction of an inch deeper in the feldspar than in the areas of quartz-feldspar intergrowth. In this somewhat elevated position, the relatively hard, tough intergrowths would be in an advantageous situation for exerting counter forces against the cutting points and edges of the bortz. As a result, the points and edges would be sheared off, and the bortz rapidly would become too smooth for further effective drilling. It is also more than likely that the bit would abrade the feldspar contained within the intergrowths slightly in advance of the associated quartz, and in doing this would continuously expose tiny, sharp bodies of quartz on the surface of the intergrowth. These small quartz projections, shaped somewhat like the teeth on a file, also would be instrumental in putting a fine polish on the surface of the bortz.

If the above conception of what takes place during the drilling of quartz-feldspar intergrowths is correct, the thought arises concerning the applicability to the problem of a bit made with very fine-grained diamond fragments in a suitable matrix. In using a bit of this type, the difference in relief between hard and soft constituents in the rocks being drilled would be almost nil, and furthermore, fresh, angular, diamond pieces would always be available for cutting until the bit was run to destruction.

Acknowledgments

Without the data, materials, and help given by members of the staff of the Longyear Company, the preparation of this paper would have been impossible. An expression of appreciation is due the Quebec Iron and Titanium Corporation for granting permission to publish information gathered during the drilling of their ilmenite property in Quebec, and also to the Winston Brothers of Minneapolis, Minn., for data pertaining to their activities near Williamsville, Mo.

* Principally personal communications from P. A. Hermiston, manager of the Canadian Longyear, Limited, Contract Drilling Division.

Safety Practices at the Crestmore Mine of the Riverside Cement Company

By R. H. WIGHTMAN* and G. H. ADAMS,* Members AIME

In order to secure good results in the prevention of accidents, it is generally recognized that the desire for such accomplishment, as well as the aggressive and constructive action to achieve it, must emanate from the Top Management and permeate successively down through the "Chart of Authority" to the workmen.

Several years ago, with the above as a basis and with full support of the Top Management, the Mining Department started a program of selling the foremen and bosses the idea that safety could be achieved and that it was desired and required. Using psychology, persistence and numerous other methods, the foremen and bosses were finally impressed with the idea that "It can be done" and "It will be done." With this in mind, new ideas were brought forth by the foremen and bosses regarding the caving operation and the specific jobs in the operation. These new ideas were taken up in round-table discussions and resulted in certain changes in the undercutting method and the actual drawing of the blocks. These changes resulted in increasing fragmentation of the blocks which prevented the usual hang-up areas in the blocks. With the rock smaller in size and within reasonable reach from the tapping drifts, accidents were prevented in the tapping section. At the same time, an enforced campaign of housekeeping throughout the mine was undertaken with very beneficial results. The foremen and bosses then, in turn, educated and trained the workers themselves to become *safety conscious*. In so doing, a constant reminder was given on all jobs assigned that the job was to be done safely as well as efficiently. No man is assigned a job who is not qualified for the type of work required. Efficiency and safety go hand in hand.

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Safety Rules

To supplement and aid this *safety conscious attitude*, certain safety rules applicable to our operation, a modified block caving system of mining limestone, in addition to the established state regulations, were put into effect and enforced. The specific rules referred to above and which will be mentioned below are in most cases simply a means of applying and complying with the very adequate "Mine Safety Orders" issued by the Industrial Accident Commission of the State of California.

Some of the safety rules and policies are as follows:

All men are required to wear safety

shoes and hard hats. Certain jobs require the use of safety glasses. Safety glasses are available to all men upon request.

On the grizzly level each tapper must keep his working place clean and clear of broken rock. When blasting, the tapper turns on the blasting signal at the end of the drift where the blasting is to be done. The signal consists of a red light and an air whistle at the entrance to the drift. The signal also is connected with a howler on the haulage level, at a point directly below where the tapper is working, to warn the trainmen of blasting above. Tappers also turn on an air valve near the chute the tapper is working. The escaping air can be heard readily in the tapping section, and serves also as a warning and as an aid in clearing the area of smoke and dust.

Another requirement which is considered to be very important, in fact so important that noncompliance is cause for discharge, is that each tapper takes a dummy fuse with him and after spitting all his fuses for the blast he spits the dummy fuse. This dummy fuse must be equal in length to the longest fuse used in the blast. Counting the shots as they go off is difficult and even though the shots can be distinguished and counted the tapper cannot return to the drift until the dummy fuse which he has carried with him has burned out. We are confident that in many cases the use of the dummy fuse has prevented men from walking back into their own blast. The tappers also are careful to notify any other tapper nearby when they are going to blast. (Cooperation among the workmen themselves is a great factor for the prevention of accidents.)

Tappers and drillers are required to cover the grizzlies with lagging before starting to drill. High or dangerous chutes are inspected by the tapper boss before the tapping driller starts to drill. Buffer timber or rail is installed in exceptionally high or wide tapping throats. Periodic inspection of the back of drifts is made by the timberman or tapping blaster.

On the haulage level, one of two-man train crew always precedes the train when backing into a haulage drift. Trainmen are required to keep a portable and detachable red light on rear car of each train. Periodic inspection of draw sets is made by the timberman.

The trolley has automatic section switches. Wooden guards are placed over the trolley at draw chute positions. There is an air whistle on the tipple approach to warn trainmen to swing in the trolley arm for proper clearance before the motor passes into the tipple. When blasting at either the tipple or pocket, the operator must throw a switch which turns on red warning lights at both places.

Also, all foremen and bosses, as well as a number of key personnel, are trained in first aid work.

Other features of our safety program include a Mine Safety Committee composed of a supervisor and mine employees. This committee makes regular monthly inspections and submits recommendations for better safety. These recommendations are taken up by a plant committee and action within 30 days taken on those finally approved.

A similar committee makes regular inspections for "good housekeeping" only; and a report of their ratings as to poor, fair, good, and excellent is posted on the bulletin board. It is felt that the effect of the report has been beneficial in creating a competitive spirit among the individual employees in keeping their respective working places in proper condition—and good housekeeping is an integral part of safety.

A mine safety bulletin board is available on which safety posters are displayed, safety slogans are written and changed periodically; with the date of the last lost time accident, name of employee injured, and the number of days since the last lost time accident.

The company encourages safety by the incentives of a monthly pay bonus to each employee in those departments completing a month without a lost time accident; a safety dinner awarded to each department that completes a year without a lost time accident; and the award of a safety medal for each supervisor whose crew completes a calendar year without a lost time accident.

Other factors that have directly or indirectly helped us in our safety program are that our mine crew has been fairly stable and our "old timers" are thoroughly trained and qualified miners, are wholeheartedly for safety and will fight for it; and they are interested enough to help train and advise the occasional new men that are taken on.

In the final analysis, a successful safety program depends principally upon the experience, common sense, attitude, cooperation, and the entire support of all individuals making up the working force. The Management takes every opportunity to compliment the employees for their effort and gives them credit for the splendid record achieved.

Sintering Characteristics of Minus Sixty-five and Twenty Mesh Magnetite

By ALAN STANLEY,* Junior Member AIME, and JOSEPH C. MEAD

Introduction

The MacIntyre Development of the National Lead Co. is located at Tahawus, N. Y. The operations involve the mining and concentrating of a titaniferous iron ore to produce an ilmenite concentrate and a magnetite concentrate.

Construction of the MacIntyre plant was commenced during the summer of 1941, when world conditions threatened to cut off the supply of Indian ilmenite. An open pit mining operation was developed and the crushing and milling equipment put in operation in July 1942. A general description of the operation was given in the Adirondack Issue of *Mining and Metallurgy* for November 1943. The metallurgy of the mill operation was described by Mr. Frank R. Milliken,* Plant Manager, National Lead Company, MacIntyre Development, and presented at the AIME New York Meeting, February 1948.

The magnetite concentrate produced in the milling operation was too fine (minus 20 mesh) to be used directly in iron blast furnace operation, and most of the magnetite had to be stockpiled in 1942 and 1943.

In 1943, the Defense Plant Corp. built a Greenawalt sintering plant at Tahawus, N. Y., to put the magnetite concentrate in a more suitable form for use in the iron blast furnace.

The Greenawalt sintering plant consists of three 10 by 25 ft sintering pans designed to produce 1800 gross tons of sinter per 24 hr. The vacuum to each pan is produced by two Greenawalt fans in series, pulling approximately 30,000 cu ft of air per minute at 50 in. water gauge vacuum. The plant started operation in August 1944. The present plant production averages 25 tons per operating pan hour (approximately 224 lb per operating hour per square foot of grate area) of plus 1 in. sinter.

Raw feed to the plant consists of 61 pct magnetite, 4 pct anthracite coal culm, and 35 pct minus 1 in. return fines which are conveyed to a pug mill where the materials are mixed thoroughly and water added to give the mixture 5.5 to 6 pct moisture. The mixed prepared feed is conveyed to two 4 by 10 ft vibrating screens where the minus 1 in. plus 5/8 in. return fines are screened out and discharged into a surge bin for use as a hearth layer. The minus 5/8 in. prepared feed is discharged into another surge bin for use as prepared feed.

A charge car, electrically operated, having a capacity of one charge of prepared feed and several charges of hearth layer, lays a thin layer of plus 5/8 in. return fines and 9 1/2 in. depth of prepared feed into the pans. A fluffing roll and a vibrator on the car fluffs and spreads the prepared feed into the pans.

An ignition car, electrically operated, ignites the top of the bed with a 30 sec flash burn. The 9 1/2 in. bed sinters in approximately 13 min. Dumping the pan, and recharging and igniting the bed requires 2 min.

To improve the quality of the ilmenite concentrate produced in the mill and to reduce the amount of titanium dioxide lost in the mill tailings and in the magnetite product, extensive research work and pilot plant operations have been done on grinding the crude ore to minus 65 mesh size (rather than to minus 20 mesh) and concentrating it by a combination of magnetic separation (for magnetite recovery) and flotation (for ilmenite recovery). These tests have proved successful in increasing ilmenite recovery and grade.

With the development of the ilmenite flotation process to a stage where a full scale flotation plant was in the design stage, the problem arose of handling the 65 mesh magnetite concentrate that would be produced. In order to study and solve the problems of handling and sintering the 65 mesh magnetite in the sinter plant, a pilot sinter plant was secured from John E. Greenawalt.

The effect of using 65 mesh magnetite in the sintering operations was then studied on the 2.4 sq ft test pan, operating under conditions as similar to the large plant as could be set up in the laboratory.

A series of tests were run in the test pan on present sinter plant feed that had been mixed in the plant pug mill. An average production and an average quality of sinter produced in this series

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* General Engineer and Sinter Plant Superintendent, respectively, MacIntyre Development, Titanium Division, National Lead Co., Tahawus, N. Y.

* Frank R. Milliken: Metallurgy at National Lead Company. TP 2355, AIME *Mining Tech.* (May 1948).

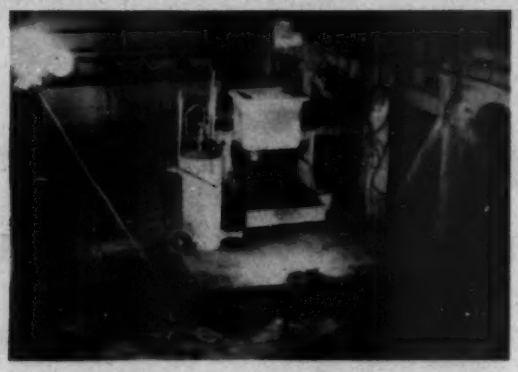

FIG 1—Sinter pan and kerosene burner.

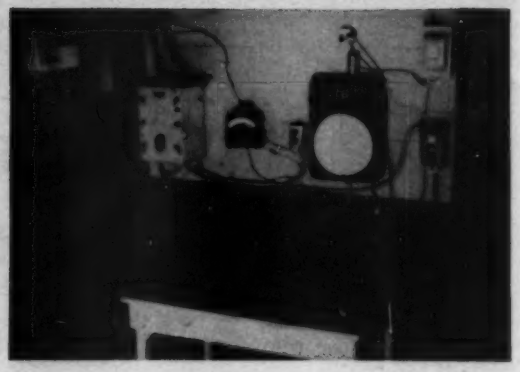

FIG 2—Instrument panel.

of tests was established as a "Standard" (8.8 lb per min) to compare the sintering of minus 20 mesh magnetite with that of minus 65 mesh magnetite.

The high points on each series of tests were repeated twice as a check on the original, to be sure that some unknown variable did not cause the change in the sintering performance. A test run of 10 pans was made on the mixture found to give the best production and quality.

Procedure

The pilot sinter plant included a sintering pan 20¾ by 16¾ in. and 9 in. deep, an ignition hood, a pressure kerosene burner for ignition, a dust collector, and a 400 cfm vacuum fan rated at 54 in. of water (see Fig 1 and 2 for instrument panel, sinter pan and kerosene burner).

Vacuum was recorded on a Bristol meter, and exhaust temperatures were read from an electrical pyrometer.

Magnetite of 65 mesh was prepared by grinding stockpile (20 mesh) magnetite, to the desired size, in a batch ball mill, and filtering in a laboratory pressure filter, Table 1.

The moisture obtained by this method of filtering was too high for pilot plant use (averaging about 9 pct) and it was necessary to dry a portion of the magnetite prior to its use in the sintering operation.

For the final "production run" tests, rod mill screen undersize (20 mesh) was ground to 65 mesh, and the magnetite was concentrated in a laboratory Crockett.

Coal was obtained from the sinter plant stockpile, and when necessary, ground in a ball mill to the desired size. After several tests had to be discarded because of poor results caused by coal with high ash content, all coal samples were submitted to the laboratory for ash determination. In order to eliminate one variable, only coal with a 10 pct or less ash content was used in the test work.

Sawdust, when needed, was obtained from local mills; limestone and slaked lime were obtained from the Chazy Lime and Stone Co.

The return fines, unless otherwise noted, were obtained by crushing plus 1 in. sinter, produced in the pilot plant, in a jaw crusher and rolls and screening it to the desired size on a laboratory vibrating screen.

Screens for screening the sinter were made locally, as were sample racks and storage bins for the raw materials. A ¼ yard gasoline concrete mixer was connected to an electric drive and used for the mixing of all batches. Balances and a hot plate were provided for moisture determinations, and an electric clock with a sweep second hand was used for timing the tests.

A group of five persons was found to be the most efficient, as continuous test work required constant work by one man grinding magnetite and coal, one man preparing sized return fines, two men preparing batches for burning, and one man completing the necessary calculations and paper work. The last three men were required when the batches were burned.

Table 1 . . . Screen Analysis of Magnetite Products

	+28	+35	+48	+65	+100	+150	+200	−200
20 mesh	8.3	16.4	19.5	14.8	12.1	9.2	5.2	14.5
65 mesh				3.0	11.2	19.8	15.8	50.2

Note: All references to 65 and 20 mesh magnetite refer to minus 65 and minus 20 mesh material.

Each batch mixed contained enough material for two tests. The charge was calculated in advance. A weighted mixture of dry and filtered magnetite was added to the mixer. This material was then mixed for 5 min and a moisture determination made. Using this moisture it was possible to calculate the exact amount of magnetite that had to be removed or added to the mixer to leave the required weight of magnetite at 5 pct moisture.

Then the remaining materials were added dry, and the necessary amount of water to provide the required moisture was sprinkled, with a sprinkling can, into the revolving mixer. When trouble with pellet formation occurred, the procedure was changed so that the necessary water was added to the return fines, and then the wet returns were added to the mixer. The latter method was more satisfactory.

Each batch, unless otherwise noted, was mixed for 5 min again.

A 15 lb bedding charge (minus 1 in. plus ½ in. sinter) was added to the pan for each burn. This gave a bedding depth of 1 in. Then the sinter pan was filled, with a weighed amount of mix, and dumped by hand through a 1 in. screen. The bed then was made level by scraping off the excess material. The weight of the mix in the pan was obtained by difference. The ignition hood was lowered into place over the pan, and the pressure kerosene burner was ignited, as the vacuum was turned on, and ignition took place for 30 sec. The hood was then raised. A record was made of the exhaust temperature as the burning proceeded and vacuum was recorded continuously with the Bristol meter. When the bedding on the grate bars glowed red (viewed by means of a mirror mounted to make the grates visible through a hole covered with glass in the bottom of the pan) the vacuum was turned off and the

sintered charge was dumped by inverting the pan.

The results of the first pan of the mix were usually poor because a cold pan did not approximate actual operating conditions. The second, or hot pan, gave results that could be compared more directly with sinter plant practice.

After each pan was burned, the material was passed over a 1 in. Ty-Rock screen three times. The material remaining as plus 1 in. was "sinter." The minus 1 in. material was passed over a ½ in. Ty-Rock screen twice, and the plus ½ in. material was "bedding." The remainder of the material (minus ½ in.) was "return fines" (note: in all references to return fines in this paper, the bedding is excluded).

In order to have a strength comparison of the sinter formed by various mixes, a "quality" figure was initiated. This figure was determined by screening the cold (the plus 1 in. sinter was allowed to air cool overnight) sinter in the same manner as above. The weights of the plus 1 in. and the plus ½ in. were combined, and this total weight was divided by the weight of the hot 1 in. sinter to obtain relative "quality."

Cycle time was figured by adding 2½ min to the burning time to allow for dumping and recharging the pan. Theoretical production in pounds per minute was determined by dividing the pounds of plus 1 in. sinter by the cycle time. In quite a few tests the amount of bedding and return fines produced was not sufficient for repeat operations. Because of this some of the plus 1 in. sinter had to be taken to provide the amounts of these materials needed. Actual production was then the weight of plus 1 in. sinter remaining after bedding and return fines requirements were met, divided by cycle time.

Discussion of Factors Investigated

The following factors were investigated during the test period:

(1) return fines, (2) coal, (3) moisture, (4) slaked lime and miscellaneous reagents, (5) oxidation, (6) limestone, (7) sawdust, (8) pellets, (9) "dust," (10) mixing time, and (11) production runs.

RETURN FINES

As far as is known, there is no sinter plant in the United States that employs sized return fines as a necessary metallurgical requirement for its operation. Mr. Worm Lund, on a visit to MacIntyre from Sweden, suggested the possibility of sized return fines in the sintering of 65 mesh magnetite.

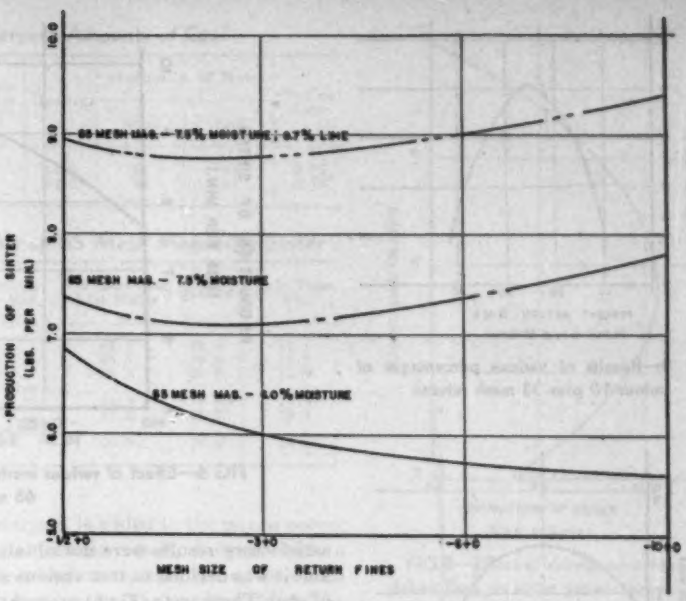

FIG 3—Results of return fines series.

In the test plant at 20 mesh, return fines were not necessary for the production of good sinter. In many cases the return fines were detrimental to test plant production. This is especially true when the returns consisted mainly of raw magnetite, coal, and dust from the dust collectors.

For sintering 65 mesh and finer magnetite, return fines are absolutely necessary for sinter production. If efficient operation and economical sinter are to be had, it is necessary to have these return fines fall within definite size ranges. The secret of sintering fine concentrates is to obtain proper porosity in the "bed" so that air can be uniformly distributed throughout and burning can progress.

A series of tests were run using minus ½ in. plus 3; minus 3 plus 6, minus 6 plus 10, minus 10 plus 20, minus 10 plus 35, and minus 20 plus 35 mesh return fines. The best results were obtained from the minus 10 plus 35 mesh tests. Because of the difficulty that would be incurred by screening to these close sizes, a series was run using minus ½ in. plus 0, minus 3 plus 0, minus 6 plus 0, and minus 10 plus 0. The results of this series can be seen in Fig 3. These tests were run with 4.0 pct coal and 25 pct return fines.

As can be seen (Fig 3) the best results were obtained using minus 10 mesh plus 0 return fines. The best percentage of this size return fines was determined by a series of tests (Fig 4). The results were satisfactory only as long as the minus 35 mesh portion did not exceed 25 pct of the total return fines weight. (This then would give a calculated minus 10 plus 35 mesh return fines figure of 18.75 pct of the total weight of the charge.)

A study of the results obtained using various amounts and sizes of return fines indicated:

1. Proper porosity is essential in producing sinter economically:
   - *a.* Too much porosity causes too rapid burning and results in poor sinter.
   - *b.* Insufficient porosity increases the burning time beyond economic limits.
2. Excess returns weaken the sinter and cause a drop in production.
3. By proper control of the size and amount of returns it should be possible to sinter magnetite finer than 65 mesh.
4. More efficient sinter plant operation could be obtained if the dust from the dust collectors and the finer portion of the returns were returned to the original source of sinter plant feed (for example: to magnetite filter).

A 20 mesh magnetite will sinter without the addition of return fines, providing adequate bedding is supplied, but blast furnace operations object to fines; this requires that the fine material be screened out and recirculated.

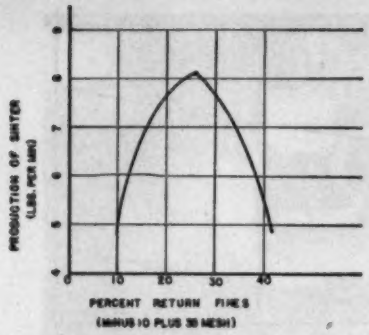

FIG 4—Results of various percentages of minus 10 plus 35 mesh returns.

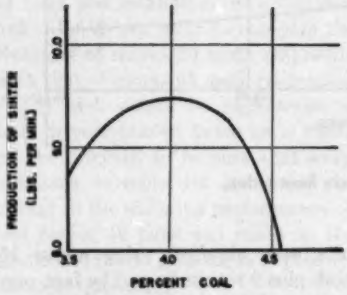

FIG 5—Effect of various percentages of anthracite culm 4 and 5 buckwheat coal on sinter production using 65 mesh magnetite.

COAL

Coal is used in MacIntyre sintering as the most economical local source of carbon.

The amount and size of coal used is an important factor in a sintering operation.

As can be seen in Fig 5, 4.0 pct coal produced the best results in sintering 65 mesh magnetite. Both greater and lesser amounts of coal tended to slow the burning time and reduce the production of sinter. Insufficient coal did not produce enough heat for proper sintering and excess coal caused too high a temperature, resulting in slagging and overoxidation with resultant poor sinter. This overoxidation was caused by an oxidizing atmosphere, and while reduction may have taken place during the initial phases of the burning, the final result was excess oxidation.

On the theory that coal used in sintering should be approximately the same particle size as the magnetite, the coal for the sintering tests was ground to 65 mesh. This fine coal, at first, seemed to help sinter production, but sawdust was being used in conjunction with it. As testing progressed, satisfactory results were not obtained, and it was decided to test various sizes of coal. These tests (Fig 6) proved that fine coal, rather than help, did much to deter good sinter production. The No. 4 and 5 buckwheat anthracite culm, as received for 20 mesh sintering, gave excellent results and made it possible to eliminate sawdust from the mix.

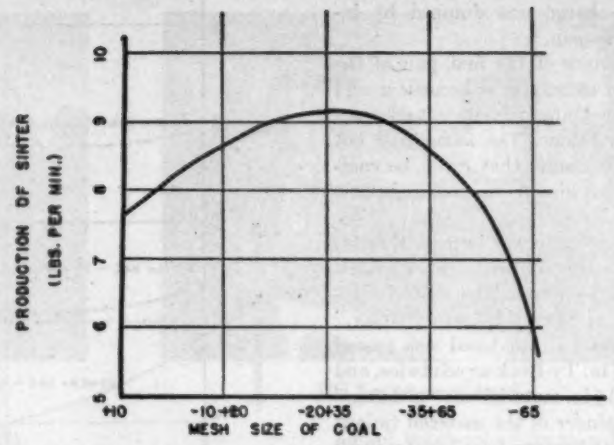

FIG 6—Effect of various mesh sizes of coal on sinter production using 65 mesh magnetite.

The fine coal tended to plug most voids, and made proper air flow impossible. The coarser coal left large voids as it burned, causing lower vacuum and faster burning time.

MOISTURE

The amount of water that can be added to a 65 mesh magnetite "mix" is limited. The "mix" will sinter, if all other proportions are correct, at a moisture between 6 and 8 pct, with the best results obtained at 7.5 pct moisture (Fig 7).

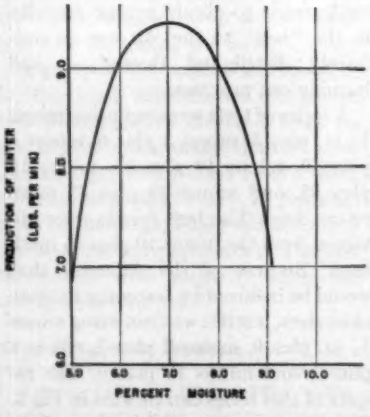

FIG 7—Effect of various percentages of moisture on sinter production with 65 mesh magnetite.

If the moisture is too high, the mix becomes semifluid and it is impossible to draw air through the bed. Thus, ignition is impossible. If the moisture is low, holes tend to form as the material is pulled through the grates. The air, following the path of least resistance, is drawn through the holes and thereby short-circuits the bed. Poor burning results and less sinter is produced.

With 20 mesh magnetite, a wider moisture range is possible. Good sinter was produced with a moisture as low as 3 pct, and as high as 8 pct. However, with 20 mesh magnetite, the percentage of moisture required is closely related to the amount of dust present in the returns. If there were more than 35 pct of minus 35 mesh material in the return fines, it was necessary to use a moisture of 7.5 pct (production with the same mix at 7.5 pct moisture was 10.8 lb per min and at 6.0 pct moisture, production was 9.6 lb per min). If the returns contain less than 35 pct minus 35 mesh material, best results are obtained with a 6 pct moisture.

SLAKED LIME AND MISCELLANEOUS REAGENTS

In an effort to reduce burning time, and thus increase production, slaked lime was added to the charge. A 10 pct increase in production was obtained with the addition of 0.7 pct lime to the 65 mesh mix (Fig 8).

If an excess of lime is added, the charge will burn hot and fast, with little sinter being produced. It is thought that the lime causes an increase in the surface tension of the water. With this higher surface tension, the water would fill less voids in the

Table 2 . . . Effect of Lime on Varying Amounts of Coal

	Production (Lb per Min)	Cycle Time (Min)	Temperature Exhaust Gas Degree C	Quality	Vacuum In. of Water		Per Cent Lime
					Start	End	
With 3 pct coal...	5.5	9.0	520	81.5	61	48	0.75
With 4 pct coal...	11.0	10.0	325	82.2	57	53	0.75

Table 3 . . . Effects of Various Reagents on 65 Mesh Magnetite Sinter

Reagent	Per Cent Reagent	Production, Lb per Min	Per Cent Plus 1 In. Sinter	Quality	Cycle Time, (Min)
Standard		9.0	65.1	78.9	13.1
Lime	0.7	9.9	63.6	78.0	11.5
Detergent	0.005	8.1	64.2	86.0	15.0
Oleic acid	0.05	(would not burn)			
Calcium chloride	2.0	9.7	61.4	74.8	12.0
Pine oil	0.05	5.1	52.2	88.2	20.5
Zinc stearate	0.5	(would not burn)			
Aluminum dust	0.5	5.8	54.8	78.9	18.0

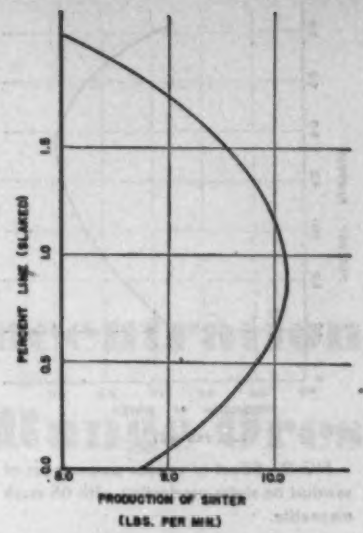

FIG 8—Effect of various percentages of slaked lime on sinter production with 65 mesh magnetite.

mix, thus leaving greater and more numerous voids than possible with unadjusted water. With more numerous voids, a greater volume of air was drawn through the bed, lowering the vacuum, decreasing the burning time, and causing a higher burning temperature. The addition of excess lime increased the surface tension to such an extent that an excess volume of air was drawn through the charge, causing such rapid and hot burning that little sinter was produced.

If the percentage of lime was held constant and the coal percentage dropped, less sinter was formed because of higher burning temperatures. Higher burning temperatures slag the sinter.

The results of Table 2 show that with 3 pct coal and 0.75 pct slaked lime, the mix burned hotter, and gave poorer results than those obtained using 4 pct coal and 0.75 pct slaked lime. This can be explained by the knowledge that a higher percentage of carbon causes excess slagging. This slagging reduces the amount of voids, and this tends to slow the burning rate. This shows, again, how critical the addition of lime would be in actual plant practice. If lime were added in excess, production would drop. If the amount of lime remained constant and the percentage of carbon dropped, production would again drop. Thus, if lime is used in actual plant practice, efficient metallurgical and quantity controls must be provided.

In order to check the surface tension theory, a series of tests were made using the following reagents (Table 3): (1) detergent ("Dreft"), (2) oleic acid, (3) calcium chloride, (4) pine oil, (5) zinc stearate, and (6) aluminum dust.

It is known that a detergent reduces the surface tension of water. Thus, if a detergent is added to the mix, a poorer sinter should result, because with a lower surface tension, the water would fill more voids. The results obtained by test work checked with this idea. When 50 g of detergent were added to the mix, the mix became "putty-like" in appearance and semifluid in nature. The sized return fines were completely disintegrated when they were placed in water containing a detergent. When this mix was added to the pan it was impossible to draw air through it, and thus it could not be ignited.

Calcium chloride, which gives a substantial increase in the surface tension of water, was next added to the mix. The calcium chloride decreased the burning time, but produced a weak sinter. The sinter cake actually expanded during burning when calcium chloride was in the mix. Why this expansion took place is unknown. Calcium chloride could not be used in sinter plant operation because of the large amounts of chlorine gas liberated during the burning; however, it was used in the pilot plant because it was known to increase the surface tension of water.

All other reagents tried gave poor results.

OXIDATION

For a short while during the test work, it was thought that the oxidation of the FeO to Fe_2O_3 was an important factor in strong sinter. To check this point many assays were made on magnetite feed, and sinter produced from this magnetite. As a result of a study of these assays it is thought that oxidation to a certain extent does produce stronger sinter by causing the iron molecules to share oxygen atoms. When oxidation progresses too far, the iron is oxidized completely and no longer shares oxygen atoms, thus producing weaker sinter. When oxidation is low, there is little sharing of oxygen atoms and again the sinter is weak.

Table 4 . . . Effects of Oxidation upon Quality of Sinter

Magnetite			Sinter			Quality
Fe	FeO	Fe_2O_3	Fe	FeO	Fe_2O_3	
57.0	31.04	46.99	56.8	21.78	57.06	75.8
56.7	31.61	45.91	56.0	16.52	61.68	87.8
58.0	32.33	46.97	56.5	9.63	70.05	72.8

Table 4 is representative of the average assays completed, and shows weaker sinter on both the high and low oxidation samples.

LIMESTONE

Because of the cheaper costs of limestone compared to slaked lime, it was thought desirable to check the effects of limestone in the "mix." The limestone was sized from minus 6 plus 10, minus 10 plus 20, minus 20 plus 35, and minus 35 plus 65 mesh. In no case did the limestone help. This was probably caused by the fact that the heat necessary to slake the lime evaporated all the moisture before the limestone was slaked, and thus the effect of the slaked lime on the water was lost.

SAWDUST

Sawdust was helpful when 65 mesh coal was used in the mix. The sawdust

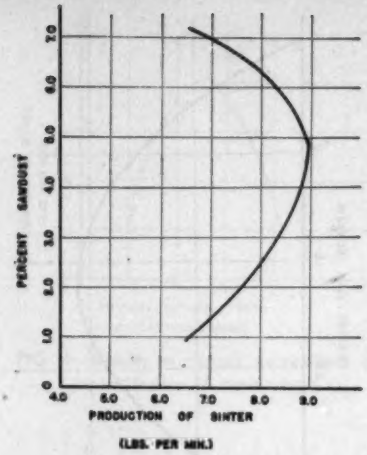

FIG 9—Effect of various percentages of sawdust on sinter production with 65 mesh magnetite.

had a tendency to "fluff" the mix in the pan, and it also made voids for air passage as the mix was burned. Fig 9 shows a high point for sawdust addition. As the percentage of sawdust was increased above 5, the mix burned very hot and fast and sinter production dropped. Sawdust elimination simplified the general problem because of the difficulties that its handling would have entailed if it were needed for 65 mesh magnetite sintering. The use of No. 4 and 5 buckwheat coal eliminated the need for sawdust.

PELLETS

A rotary mixer was used for pilot plant mixing. During the mixing of some batches, pellets were formed by magnetite building up around particles of return fines. The coarser the return fines, the larger the pellets would become. These pellets interferred with sinter production by causing short-circuiting of air around them because they had no porosity. Therefore they were unable to be sintered. The pellets were baked by the heat around them, but crumbled when subjected to the slightest pressure. No figures were obtained and the only method of making these determinations was by visible observation.

In sinter plant operation, pug mills are used for mixing and this problem should not occur. However, if pellets do form, some method to eliminate them must be provided. This does not infer sintering of a completely pelletized charge was undertaken. In all cases the pellets were mixed with unpelletized material, and under such conditions, were detrimental.

DUST

Dust is defined in this test work as material in the return fines, less than 65 mesh for 20 mesh magnetite sinter and minus 200 mesh for 65 mesh magnetite sinter. After it was known that excess fine material would reduce sinter production with 65 mesh material, tests were made to determine what effects "dust" would have on minus 20 mesh material. It was found that if the moisture was increased as the percentage of "dust" increased, the amount of "dust" in the "mix" had little effect on sintering of 20 mesh magnetite.

MIXING

Early in the project, several mixing tests were run using pug mill discharge. When this material was further mixed in the rotary mixer, a sharp production increase was noted. This mixing was checked again at the end of the project, and no increase in production could be attained by further mixing. In some cases production fell off after additional mixing, probably caused by water filling the voids in the return fines. It is thought that the pug mill blades, and other plant conditions were responsible for the poor results obtained during the first testing, but with the pug mills in good shape, as far as blade wear is concerned, a complete mix should be attainable. It was found that a 5 min rotary mix, after the magnetite had been brought to the correct moisture, gave the best results for rotary mixing in the test plant.

PRODUCTION RUNS

When the test work was completed it was necessary to test the best results. For this purpose, several pans were burned in rapid succession to see how the "mix" would react in a hot pan over a period of time. The recommendations made were on the basis of the production run results.

Conclusions

The best quality and quantity of sinter on this test pan were obtained with the following mix:

4.0 pct coal
25.0 pct return fines (minus 10 mesh plus 0) (at least 18.75 pct minus 10 mesh plus 35 mesh)
70.3 pct magnetite (minus 65 mesh size)
0.7 pct slaked lime.

The average production on the test pan with the above "mix" was 9.9 lb per min or 246 lb per hr per sq ft of grate area. This is slightly higher than average production in the sintering plant pans on minus 20 mesh magnetite.

The pilot plant work indicates that minus 65 magnetite can be sintered in the sinter plant if close control can be maintained over the following items:

1. Adequate amount of bedding material (minus 1 in. plus ½ in. return fines for hearth layer.
2. Sized return fines (minus 10 mesh plus 0).
3. Anthracite coal (size and moisture percentage).
4. Moisture percentage in the "mix."
5. Magnetite (moisture percentage).
6. Percentage of magnetite, coal, lime, and return fines in the prepared feed to the plant.

The results obtained in the sinter testing show that a more accurate metallurgical control on overall operation will be necessary for sintering minus 65 mesh magnetite than is needed in present plant operation for sintering minus 20 mesh magnetite.

Before the present sintering plant can handle minus 65 mesh magnetite, the plant flowsheet will have to be changed to include the conveying, cooling, screening, storing, and weighing of both return fines and bedding material.

Because of the difference between the size of the test pan (2.4 sq ft grate area) further development work will be needed in the sintering plant. A production run of at least two months would be required on minus 65 mesh magnetite to study the effect of the finer mesh size magnetite on the operation and maintenance of the plant.

Acknowledgment

The authors wish to thank the following individuals for their suggestions and help in making the sintering tests and for their help in making it possible for them to write this paper: Messrs. Gloyd M. Wiles, Chairman of the Mining Committee, National Lead Co., Frank R. Milliken, Plant Manager, MacIntyre Development, National Lead Co., A. R. Reiser, former Assistant Plant Manager, M. F. Morgan, Arthur G. McKee & Co., John E. Greenawalt, and the following metallurgical personnel of MacIntyre Development; Messrs. Ernest M. Spokes and Elton Geist.

Humphreys Spiral Concentration on Mesabi Range Ores

By WHITMAN E. BROWN,* Junior Member AIME, and LOUIS J. ERCK†

The installation in 1948 of a Humphreys spiral concentrator section at the Hill-Trumbull plant of The Cleveland-Cliffs Iron Co. is the latest commercial method on the Mesabi Range being used for the recovery of fine iron ore. In two stages of concentration 84 spirals treat approximately 120 long tons per hour of minus ½ in. ore. These spirals augment the production of the heavy-density plant which recovers the plus ½ in. iron from the plant crude ore. Structure of the ore is such that, when crushed, about 50 pct of the plant crude is minus ½ in. size. After desliming in a 66 in. Akins classifier, grinding the classifier product in a ball mill and again desliming in a 78 in. Akins classifier about 15 pct of the crude ore remains to be treated in the spiral plant.

Overall recovery of the ball mill feed in the Hill-Trumbull spiral plant during the 1948 season was about 53 pct of the weight and 66 pct of the total iron content. Of the actual feed delivered to the spirals from the 78 in. classifier 64.9 pct of the weight was recovered containing 73.7 pct of the total iron. The average analysis of the spiral concentrate is 55.05 pct Fe and 14.83 pct SiO_2.

An average of mill assays during the 1948 season is given in Table 1.

Table 1 . . . Average of Mill Assays during 1948

Product	Assays, Pct	
	Fe	SiO_2
Classifier product (ball mill feed)	44.20	31.47
Spiral feed (deslimed ball mill product)	48.50	24.84
Spiral concentrate	55.05	14.83
Spiral tailing	36.39	43.01

The Hill-Trumbull plants treat both wash ore and jig-type ore. During the 1948 season spirals were used only on the fine sizes of jig-type ore. At times the deslimed minus ½ in. size is of sufficient grade to warrant bypassing the spirals but in the main the fine size is delivered to the 48 first stage spirals and thence to the 36 cleaner or second stage spirals.

In the past several years numerous Mesabi Range ores in the fine size category have been tested by a wide variety of beneficiating equipment. The prodigious amount of research devoted to the recovery of values in the fine ore reflects the dire need of the iron industry to squeeze the last possible unit of iron from the mined ore and to develop suitable equipment and processes to satisfactorily concentrate the leaner horizons as they are encountered in the process of mining. Large governmental and private appropriations have been allotted, new laboratories built, research and technical staffs increased, and operator-manufacturer cooperative agreements applied to hasten the solution of the problem.

The University of Minnesota, at its Eighth Annual Mining Symposium in January 1947, dedicated as its objective the bringing to light of technological developments in iron ore beneficiation. Papers were given at that time relating to processes being developed. Specifically, these papers reviewed flotation, Baum jig, selective media concentrator, Hardinge concentrator, Dorrco sizer, heavy media concentration, Hydrotator, and the Humphreys spiral.

The Cleveland-Cliffs Iron Co., after encouraging results from tests in a small pilot plant using one full size spiral, made the first commercial installation of Humphreys spirals on the Mesabi Range at their Hill-Trumbull mine, Taconite, Minn. This company

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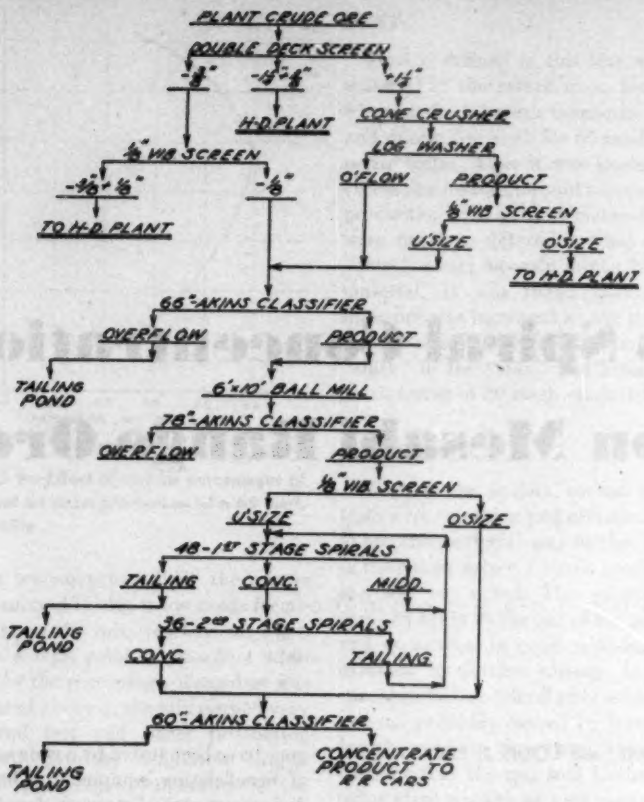

FIG 1—Flowsheet of fine ore section Hill-Trumbull plant.

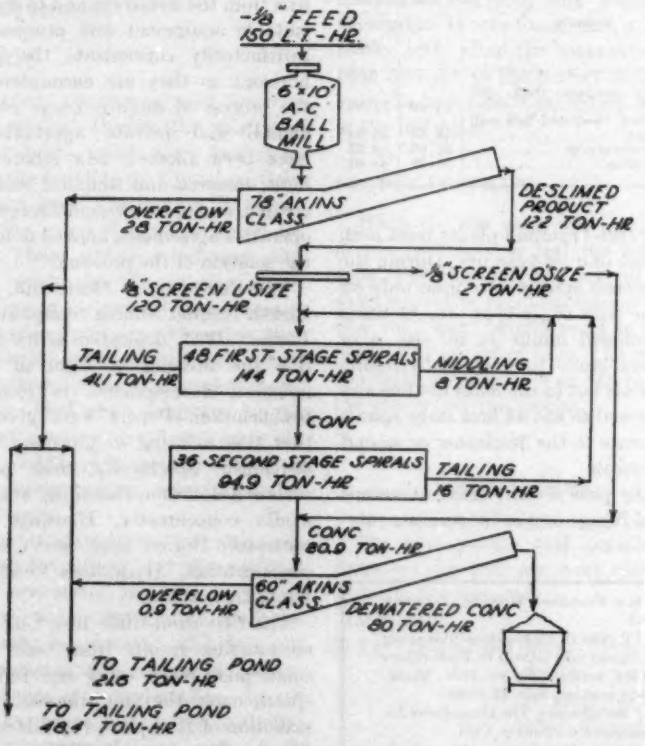

FIG 2—Flowsheet of solids distribution in spiral plant.

had tried in the laboratory and commercially, several other methods of concentration on the fine ores representative of the Hill-Trumbull group. These other methods of concentration failed to produce consistently an acceptable grade and recovery of finished product. Such failure was largely because of the inability to reject silica and interfering middling in the coarser sizes. This is particularly true of any method of concentration, either free-settling or hindered, in which a vertical component is used for eliminating the middling fraction. Success of the spiral is partly because of the centrifugal force with a horizontal component, resulting in substantial elimination of the coarser middling and tailing fractions while treating a wide range of sizes.

The Cleveland-Cliffs Iron Company Plant

A general flowsheet of the plant treating the fine sizes of Hill-Trumbull jig-type ore is shown in Fig 1. Flowsheet of solids distribution in the spiral plant is shown in Fig 2. Preparation of ore for the heavy-density plant is near standard practice. A slight divergence begins where the minus ⅛ in. size material leaves the 66 in. Akins classifier as a product and is routed to a 6 by 10 ft Allis-Chalmers ball mill. The ball mill operates in open circuit, is charged with 2 in. diam balls, driven by a 200 hp motor and has a throughput of approximately 150 long tons of solids per hour. Pulp density in the mill is approximately 65 pct solids. The purpose of the ball mill is to subject the ore to a differential grind which liberates silica from the iron ore particles, and middling from the ore particles and from silica. It also assists in breaking down porous iron oxide particles. The ground product flows into a 78 in. Akins classifier which makes a size separation at nearly 200 mesh. The slime or overflow waste analyzes about 25 to 30 pct Fe and 50 to 60 pct SiO_2. Classifier product containing about 75 pct solids drops onto a 4 by 8 ft. Allis-Chalmers Low-Head vibrating screen. The screen cloth has ⅛ by 2¾ in. slotted openings and is set with the long dimension of the slot parallel to the flow of the pulp. Experience has proved that as a general rule the plus ⅛ in. particles, after the abrasion grind, are of acceptable grade for final plant concentrates. Average analysis of the screen oversize was 54.20 pct Fe and 15.24 pct SiO_2. Primarily, the pur-

Table 2 . . . Power Distribution in Spiral Plant

Item	Connected Horsepower	Hp-hr per Ton of Feed to Ball Mill
Ball mill (6 by 10 ft)	200	1.333
78-in. Akins classifier (desliming ball mill product)	20	0.133
Vibrating screen (4 by 8 ft)	5	0.033
First stage feed pump (8 by 6 in.)	40	0.266
Second stage feed pump (8 by 6 in.)	25	0.165
Concentrate pump (5 by 4 in.)	30	0.198
Tailing pump (10 by 8 in.)	50	0.333
60-in. Akins classifier (dewatering concentrate)	7.5	0.050
84 Humphreys spirals	0	0.000
Total	377.5	2.511

Akins classifiers by Colorado Iron Works Co. Denver, Colo.
Ball mill, screen and pumps by Allis-Chalmers Manufacturing Co., Milwaukee, Wis.
The largest single power requirement is charged against the ball mill but preliminary spiral test results indicated a differential grind was justified for good recovery and separation.

pose of the screen was for removal of tramp oversize to prevent plugging pipelines and ports in the spiral installation. Usually the amount of screen oversize was a small percentage of the total feed and did not influence seriously the grade of the final concentrate.

Screen undersize flows to a sump which collects feed for the 48 first stage spirals. These spirals produce a finished tailing, middling, and a rougher concentrate. Middling is returned to the first stage feed sump. Rougher concentrate drops into another sump which collects the feed for the 36 second stage spirals. The second stage spirals produce a finished concentrate and a tailing. This tailing is returned to the first stage feed sump. A circulating middling is thereby accumulated in the rougher spirals consisting of the first stage middling and the second stage tailing amounting to about 15 pct of the new feed. Finished spiral concentrate joins the ⅛ in. screen oversize and is pumped to a 60 in. Akins classifier for dewatering prior to dumping into railroad cars. The heavy-density plant concentrates are dumped directly into the railroad cars.

Spiral Plant Operation and Performance

POWER

Table 2 shows distribution of power requirements associated with this spiral installation.

As operated during the 1948 season and based on the 150 long tons of feed per hour, power requirements for material handling in the spiral section were 2.51 hp-hr per ton of concentrate. This is equivalent to 0.40 tons new feed per horsepower-hour and 0.21 tons finished concentrate per horsepower-hour. The only power used in the actual concentration is in the 78 in. Akins classifier; the remaining power is used in grinding and material handling.

LABOR

All 84 spirals are policed by one operator who removes extraneous matter from the spirals, adjusts pulp density of spiral feed, and adjusts splitters and quantity of wash water. He also has time for additional duties including sampling and tending to the ball mill, pumps, classifiers, and screen.

If there is a power failure the spirals empty themselves by gravity.

Spiral operators are easily trained and after a reasonable length of time can become proficient. They are required to observe structure of the ore and to note changes which would suddenly increase or decrease the pulp density. When such changes occur, water is added or decreased to maintain the proper pulp density and launder make-up water. Occasionally, the Hill-Trumbull plant crude ore does not provide a sufficient amount of minus ⅛ in. material to adequately feed the spiral section. Before ringing for more feed the spiral operator consults the heavy-density plant foreman to determine how much more feed his section can stand. If the heavy-density plant is operating at maximum capacity and there is not enough ore coming to the spirals, this material is pumped to the 60 in. Akins classifier and loaded into the railroad cars without going to the spirals. The heavy-density concentrates then absorb the difference in grade to make a reasonably satisfactory final mill concentrate. The other extreme has occurred in which the plant crude consisted of too much minus ⅛ in. size material to treat in the spirals and the operator requests less feed. Fortunately these extremes are rare. If the tendency of the feed is to contain an abnormal amount of the upper size ranges, the pulp density is adjusted to about 45 pct solids in order to recover the coarse valuable particles. Conversely, if the feed tends to contain abnormal amounts of the finer sizes the pulp density is decreased to recover the finer sizes. By visual observation of the spiral tailing a good operator can readily determine adjustments needed.

Table 3 . . . General Operating Data

Long tons new feed per rougher spiral per hr	2.50
Long tons new feed per spiral per hr	1.43
Long tons new feed per day per sq ft floor space (Includes only spirals which occupy 1163 sq ft of floor space for roughers and cleaners.)	2.47
Size of feed, in. (8 pct minus 200 mesh)	minus ⅛
Wash water per spiral, gpm	5 to 7
Wash water total, gpm	420 to 588
Solids in feed, pct	30 to 45
Pulp feed, gpm	20 to 30
Circulating load, pct	10 to 20
Ratio of concentration, overall	1.89:1
Ratio of concentration, spirals	1.25:1
Total water used, gpm	2,600
Total water used per ton of feed (150 ton basis), gal (Includes water lost in 78 in. Akins classifier overflow, ⅛ in. screen spray water, launder make-up water, pump water, and spiral wash water.)	1,040

	Average Assays, Per Cent	
	Fe	SiO_2
Average 66 in. Akins classifier product (ball mill feed)	44.20	31.47
Spiral feed (deslimed abrasion mill product)	42.50	24.84
Spiral concentrate	55.05	14.83
Spiral tailing	36.39	43.01
Weight recovery (spirals only)	64.9	
Fe recovery (spirals only)	73.7	

Table 4 . . . Monthly Summary of Spiral Performance Assays, Per Cent

Product	May		June		July		August		September		October		Average	
	Fe	SiO_2	Fe	SiO_2	Fe	SiO_2	Fe	SiO_2	Fe	SiO_2	Fe	SiO_2	Fe	SiO_2
Ball mill feed	41.47	35.12	46.36	28.21	43.41	32.16	45.67	29.67	42.68	33.89	42.40	34.75	44.20	31.47
Spiral feed	42.62	33.29	49.30	23.31	48.43	24.62	48.48	25.29	48.69	24.94	48.31	25.41	48.50	24.84
Spiral concentrate	51.45	19.71	54.98	14.67	54.55	15.35	53.94	17.05	55.60	13.04	56.10	13.26	55.05	14.83
Spiral tailing	37.88	40.42	41.11	35.65	35.11	44.73	34.45	46.33	33.50	47.79	33.65	47.07	36.39	43.01
Weight recovery, pct	34.94		59.04		68.87		71.98		65.76		65.30		64.90	
Iron recovery, pct	42.18		65.84		79.49		80.10		76.45		73.83		73.67	

That portion of the ball mill product overflowed from the 78 in. Akins classifier is not shown above but averaged about 25 pct Fe and 60 pct SiO_2. Note the progress made in the technique of operating the spiral installation. Excluding May, the spiral feed in general is consistent, while the spiral concentrate improves in grade without sacrificing additional iron in the tailing.

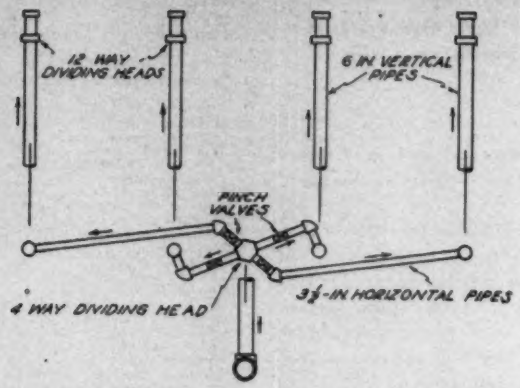

FIG 3—Feed distribution to four banks of first stage spirals.

FIG 4—Part of the 48 first stage 5-turn spirals. Hill-Trumbull spiral plant of the Cleveland-Cliffs Iron Co.

Table 5 . . . Screen Analysis of Ball Mill Feed and Product

Size	Ball Mill Feed				Ball Mill Product			
	Wt, Pct	Cum Wt, Pct	Assays, Pct		Wt Pct	Cum Wt, Pct	Assays, Pct	
			Fe	SiO_2			Fe	SiO_2
+0.371 in.					0.52	0.52	56.8	10.8
−0.371 in. + 3 mesh	0.09	0.09			0.43	0.95	44.6	30.2
−3 + 6	2.36	2.45	52.3	18.4	1.76	2.71	53.4	15.5
−6 + 8	15.14	17.59	47.4	26.0	2.65	5.36	52.4	16.8
−8 + 14	23.66	41.25	46.5	27.8	8.96	14.32	50.2	22.1
−14 + 28	21.56	62.81	42.4	34.5	16.42	30.74	46.3	28.2
−28 + 35	11.13	73.94	39.9	38.7	10.30	41.04	42.3	34.1
−35 + 48	7.71	81.65	40.4	37.8	11.17	52.21	41.5	36.0
−48 + 65	6.15	87.80	41.5	36.4	9.00	61.21	41.8	36.1
−65 + 100	3.11	90.91	40.1	39.1	7.18	68.39	40.1	38.9
−100 + 150	2.41	93.32	32.1	51.5	4.63	73.02	35.2	45.9
−150 + 200	2.56	95.88	19.0	70.2	4.11	77.13	29.0	55.6
−200	4.12	100.00	17.4	72.7	22.87	100.00	35.1	54.3
Total	100.00	100.00	41.5	35.1	100.00	100.00	41.4	37.9

When the structure of the ore is such that the deslimed minus ⅛ in. ore reaching the spirals is of acceptable shipping grade it is sent directly to the concentrate thickener and the spirals are bypassed.

Data pertinent to spiral operation representative of the early part of the season of 1948 are given in Table 3. Average results by months are shown in Table 4.

GRINDING

The 6 by 10 ft Allis-Chalmers ball mill was fed 150 long tons of solids per hour. Preliminary laboratory tests on this ore showed a distinct advantage in both grade and recovery if the fine ore were given a differential grind. Grinding is fast and in open circuit. Table 5 shows the size distribution before and after abrasion grinding giving analysis of samples from a single shift and illustrating results obtainable from abrasion grinding of this ore. The table shows that at about 35 mesh the finer sizes begin to appear in larger weight percentages in the product than in the feed. Iron content also goes up in the finer sizes in the ball mill product indicating some grinding action on the locked iron and silica as well as on the silica alone. Silica content in the coarse sizes has been materially lowered as far down as 35 mesh. The bulk of the grinding work appears to be on the minus 6 plus 28 mesh sizes.

CLASSIFICATION

The 78 in. Akins classifier which receives the ball mill discharge helps concentrate the ore to a slight degree by desliming at about 200 mesh. Classifier overflow contains about 80 to 90 pct of minus 200 mesh solids and the product about 6 to 7 pct of minus 200 mesh solids. The overflow rarely contains more than 30 pct Fe.

SCREENING

Nature of the ore is such that the plus ⅛ in. particles after the abrasion grind are usually of sufficient grade to consider concentrate. The plus ⅛ in. particles come from spillage and screen cloth wear in other parts of the mill ahead of the spiral section. Screen oversize is usually just a few tons per hour and if an excess amount appears the spiral operator notifies the foreman so that the condition may be corrected. The screen cloth, 4 ft wide by 8 ft long and containing ⅛ by 2¾ in. slots, was tried with the long dimension across the line of pulp flow and parallel with the flow. When the slots were placed across the flow, screening was highly inefficient and it was nearly impossible to get good size separation at a rate of 120 tph. Turning the slots parallel to the flow immediately permitted easy and effective separation. A slotted screen is not the most desirable type to use for preparing a spiral feed. Flat ore particles are difficult to recover in the spiral as such particles when caught

by the fast water in the spiral tend to remain there and are discharged as tailing. A granular or blocky type of feed is more satisfactory unless, of course, the flat particles are gangue. Screen analysis of the ball mill product shows that the coarse sizes have a higher iron content than the fine sizes.

FEED DISTRIBUTION

To first stage spirals:

On leaving the first stage feed pump the pulp enters an 8 in. vertical pipe. A Humphreys dividing head with four horizontal 3½ in. pipe size outlets is located on top of the pipe. The outlets are branch lines, each containing the pulp feed for one bank of 12 spirals. These branch lines are horizontal 3½ in. pipes with pinch valves for closely equalizing and regulating the pulp flow to each bank of spirals. A 6 in. vertical pipe is attached to the end of each horizontal line and atop each 6 in. pipe is a 12-way Humphreys dividing head with 1¼ in. side outlets. Each 1¼ in. outlet provides feed for one spiral. Fig 3 and 4 illustrate the piping used in the distribution system of the first stage spirals.

To second stage spirals:

Pulp leaving the second stage pump enters a vertical 8 in. pipe. On top of the 8 in. pipe is a tee which is reduced to two 4 in. horizontal outlet pipes, splitting the feed evenly two ways. At the end of each horizontal outlet a vertical 6 in. pipe rises to an 18-way Humphreys dividing head with 1¼ in. side outlets. Each 1¼ in. outlet provides feed for one spiral. Fig 5 and 6 illustrate the piping system for feeding the 36 second stage spirals.

Spiral Concentration

Structure of the ore to be concentrated affects grade and recovery of spiral products. At the Hill-Trumbull plant two types of ore are treated in the spirals:

1. Hard dense hematite with very little limonite and at times containing some taconite. Very fine grinding would be required for complete liberation of silica from the iron oxide. The concentrate is characterized by a high silica to iron ratio. The plus 14 mesh sizes of this type ore contain many flat pieces of iron oxide while the silica is rounded or blocky.

2. Hard and soft hematite and limonite mixed with varying amounts of paint rock. Contained silica is liberated more easily from the iron oxide particles than in the other type ore. Spiral concentrate from this type ore usually contains less iron than the first type, but the silica content is lower. The reason for this is attributed to limonite in the concentrate.

It is well-known that the structure of the ore varies widely, but screen analyses are not available to illustrate the extent of this variation. Table 6 shows screen analyses of spiral products during one shift in the early part of the season. Fig 7 is a graph showing spiral performance during this period. Good recovery is shown in all sizes down to 200 mesh. Minus 200 mesh particles require a greater degree of cleaning than the coarse particles. Also the nature of spiral action is such that the extreme fines are difficult to recover. Good grade and recovery from these ores is obtained down to plus 28 mesh at which point the grade decreases and the recovery increases. Recovery begins to drop off in the minus 150 mesh sizes but the cleaning is fairly good.

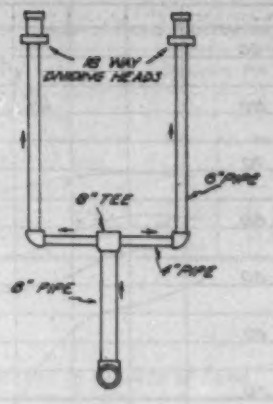

FIG 5—Feed distribution to two banks of second stage spirals.

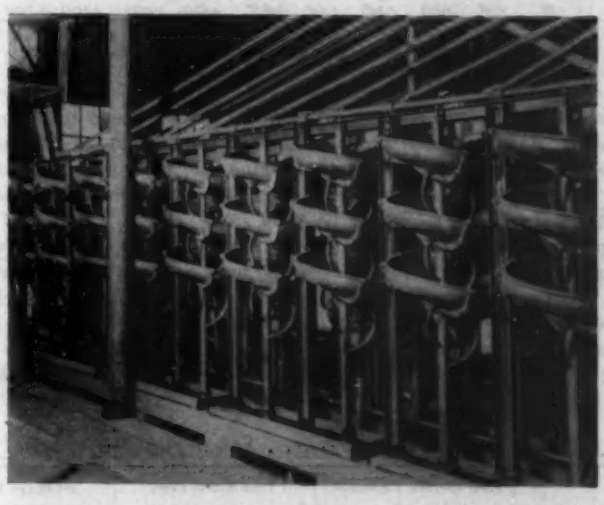

FIG 6—Part of the 36 second stage 3-turn spirals. Hill-Trumbull spiral plant of The Cleveland-Cliffs Iron Co.

In spiral concentration of an ore in which a large percentage of the weight and values is in the plus 35 mesh size, it is usually necessary to operate with a feed pulp density of about 40 pct solids and a pulp rate of 25 to 30 gpm. Both of these factors work against recovery of fine size particles. The high density is necessary to force the coarse iron oxide particles out of the stream and into the concentrate ports. The high pulp rate keeps the coarse gangue, fine gangue, and part of the fine iron suspended in the spiral stream.

The first stage concentrate indicates good grade would be obtained if all the plus 35 mesh sizes could be removed as a finished product. This can be done, in part, as a large portion of the coarser sizes are removed in the upper turns of the spiral. Removal of a finished concentrate in the first stage would be desirable for several reasons:

1. It is a good policy to remove a finished product as soon as possible. Second stage tailing contains some coarse iron recovered in the first stage. This tailing is returned to the first stage feed as part of a circulating load and hence is subject to possible loss in the tailing of the first stage operation.

2. The load of the second stage spirals would be lightened.

3. Circulating load would be decreased and the overall capacity of the spirals could be increased.

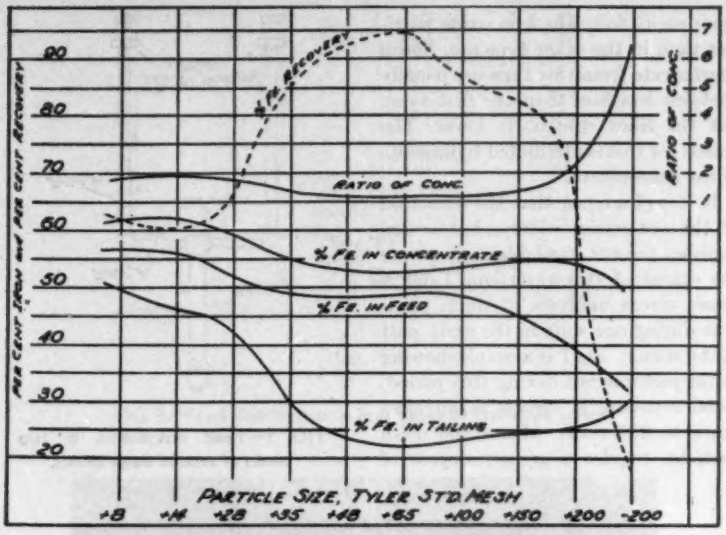

FIG 7—Distribution of particle size in spiral products.

4. Early removal of the coarse sizes would produce a closer sized product for the second stage feed. This would simplify spiral adjustments in the second stage and furnish a cleaner or higher grade product in the 28 to 65 mesh size range.

These items are under consideration as a possible flowsheet change.

It would appear from the screen analysis given in Table 6 that too wide a cut was made in the first stage on the intermediate sizes resulting in taking into the concentrate and middling too much silica and obtaining a low grade product. The zone of concentration is not sharp between the heavy and light minerals especially when the ore contains iron oxide particles with considerable range of specific gravity. Therefore it would seem better to keep the concentrate ports open rather wide to insure recovery of both coarse and intermediate sizes in the first stage. Limonite, paint rock, ferruginous silica, true middling, and porous iron particles having a specific gravity near 3.5 to 3.8 tend to intermingle with silica in the zone of concentration. If too narrow a cut is made the tailing is high in iron and the concentrate is not necessarily compensated or proportionately raised in grade. Removal of plus 28 mesh after the first stage of concentration would prepare a better suited feed for second stage concentration. Studies are being made to determine the best method for this removal.

An example of spiral results obtained when the grade of feed fluctuates widely between shifts is illustrated in Table 7.

Mill reports give no explanation for the high tailing obtained during the 11–7 shift although it would appear that the proper adjustments had been made on the 3–11 shift to compensate for the lower grade feed and then during the 11–7 shift when the grade of the feed increased, no adjustments were made. Such adjustments are necessary or more critical in ores of low concentration ratio, such as iron, manganese and chrome, than in other ores of higher concentration ratio because of the large weight percentage that needs to be removed from the ore

Table 6 . . . Screen Analyses of Spiral Products, Per Cent

Size	Feed (Screen Undersize)			True Spiral Feed			1st Stage Tailing			2nd Stage Concentrate			Recovery		
	Wt	Fe	SiO_2	Wt	Fe	SiO_2	Wt	Fe	SiO_2	Wt	Fe	SiO_2	Total Wt	Wt in Size	Fe in Size
+0.371 in.															
−0.371 in. + 3 mesh															
−3 + 6	0.95	60.20	9.20	0.19			0.18			0.74	62.80	6.30	0.56		
−6 + 8	3.68	58.30	12.10	2.85	58.90	10.90	5.00	48.40	25.50	2.87	62.90	5.80	2.19	68.3	73.69
−8 + 14	12.46	57.90	13.30	11.86	56.60	13.60	17.02	44.70	31.70	9.96	63.60	5.50	7.59	69.8	76.67
−14 + 28	21.90	52.10	21.40	20.98	51.50	21.40	22.30	44.50	44.50	18.85	59.00	12.30	14.38	52.4	59.34
−28 + 35	15.00	49.20	25.80	14.83	47.60	27.20	7.86	27.00	58.50	17.05	53.00	20.20	13.00	85.4	92.00
−35 + 48	15.03	50.10	25.05	13.22	47.50	27.70	3.98	21.10	67.40	18.14	52.20	21.80	13.83	93.5	97.42
−48 + 65	8.30	50.80	23.80	10.00	47.70	27.40	2.34	19.80	69.30	13.05	53.60	20.50	9.96	91.7	96.75
−65 + 100	8.10	51.50	23.25	7.44	48.00	27.70	2.96	22.30	65.90	10.02	55.10	18.30	7.64	89.0	95.22
−100 + 150	4.25	50.20	24.95	4.56	45.80	30.20	3.22	25.10	61.90	4.83	56.40	16.70	3.68	80.2	90.11
−150 + 200	3.56	44.80	33.10	3.33	40.80	37.70	4.96	24.10	63.20	2.82	56.40	16.90	2.15	64.1	80.70
−200	6.77	30.60	53.70	10.74	32.80	49.70	30.18	28.80	56.30	1.67	48.50	28.40	1.27	9.1	14.42
Total	100.00	50.44	24.24	100.00	47.96	26.90	100.00	33.01	49.61	100.00	55.87	16.74	76.25		84.45

Size	1st Stage Middling			2nd Stage Tailing			1st Stage Concentrate			60-in. Akins Overflow		
	Wt	Fe	SiO_2	Wt	Fe	SiO_2	Wt	Fe	SiO_2	Wt	Fe	SiO_2
+0.371 in.												
−0.371 in. + 3 mesh												
−3 + 6	0.07			0.36	50.70	22.00	0.75	62.20	6.70			
−6 + 8	1.07	55.70	15.20	1.82	50.80	22.10	2.74	62.00	7.00			
−8 + 14	8.75	58.30	11.10	14.75	54.00	17.60	11.83	62.00	7.20			
−14 + 28	22.06	53.30	18.80	33.63	47.30	27.70	21.92	55.60	16.30			
−28 + 35	16.57	43.50	33.30	16.13	38.70	41.10	16.41	50.30	24.00	0.05		
−35 + 48	16.12	37.40	43.00	10.74	34.10	48.20	16.47	50.10	24.45	0.30		
−48 + 65	12.10	35.30	46.50	5.99	38.20	49.50	11.97	51.70	22.75	1.89	31.30	52.40
−65 + 100	11.30	37.80	43.40	4.67	32.10	51.90	8.59	52.60	21.50	6.62	33.00	49.80
−100 + 150	3.83	41.50	37.90	2.46	32.20	57.70	3.84	53.00	21.30	10.57	37.30	43.70
−150 + 200	4.72	45.20	32.30	3.38	35.10	47.70	2.89	51.20	23.60	14.24	36.70	44.90
−200	3.41	42.30	36.70	6.06	31.20	53.60	2.59	40.20	39.65	66.33	27.60	58.15
Total	100.00	44.44	32.50	100.00	42.24	35.87	100.00	53.46	19.73	100.00	30.36	54.05

(Tables 8 and 9).

Example of shifts showing minimum and maximum recovery, not consecutive shifts, are given in Table 10.

Low recovery shown in Table 10 is probably due to early season inexperience in operation of the spiral plant and an overall average shows that as the season progressed the metallurgical results were increasingly better.

Summary

Use of the Humphreys spiral concentrator for treating minus ⅛ in. iron ore to produce an acceptable grade concentrate has been adopted and operated to the satisfaction of The Cleveland-Cliffs Iron Co. at the Hill-Trumbull plant on the Mesabi Range.

The spiral section of the plant, during the 1948 season, concentrated jig-type or retreat ores at an average rate of 120 long tons per hour and recovered about 78 tph of concentrate. Average grade of concentrate in October as the season ended was 56.10 pct Fe and 13.26 pct SiO_2.

Spiral metallurgy was satisfactory and it is believed that even better results can be obtained. Studies are being made for possible future changes.

The Cleveland-Cliffs Iron Co. has shown its desire to further the technology of iron ore beneficiation by its willingness to try new methods.

Acknowledgments

The authors are indebted to the entire organization of The Cleveland-Cliffs Iron Co. for its whole-hearted cooperation and especially to Messrs. W. A. Sterling, George Beasley, W. Van Slyke, Jack Chisholm, and William Johnson for their close cooperation and interest in making a success of the operation of this new method of iron ore beneficiation.

Table 7 . . . Spiral Results Obtained when Grade of Feed Fluctuates

Shift	Assays, Pct						Recovery, Pct	
	Feed		Concentrate		Tailing		Wt	Fe
	Fe	SiO_2	Fe	SiO_2	Fe	SiO_2		
7–3	51.80	18.50	58.50	7.85	34.30	45.60	72.3	81.4
3–11	41.40	38.30	54.60	14.10	32.60	48.70	41.6	54.8
11–7	53.20	16.10	57.50	9.70	46.00	27.00	62.6	67.6

Table 8 . . . Example of Minimum Variation in Quality of Feed

Shift	Assays, Pct						Recovery, Pct	
	Feed		Concentrate		Tailing		Wt	Fe
	Fe	SiO_2	Fe	SiO_2	Fe	SiO_2		
11–7	53.0	20.10	56.70	14.30	34.20	47.40	83.50	89.30
7–3	52.7	21.40	56.70	15.20	36.50	44.40	80.20	86.30

Table 9 . . . Example of Results Obtained from Low Grade Feed and High Grade Feed, Not Consecutive Shifts

Assays, Pct						Recovery, Pct	
Feed		Concentrate		Tailing		Wt	Fe
Fe	SiO_2	Fe	SiO_2	Fe	SiO_2		
35.7	42.30	51.10	17.65	23.00	62.30	45.2	64.7
56.7	16.00	59.50	12.50	41.80	36.60	84.2	88.4

Table 10 . . . Example of Shifts Showing Minimum and Maximum Recovery

	Assays, Pct						Recovery, Pct	
	Feed		Concentrate		Tailing		Wt	Fe
	Fe	SiO_2	Fe	SiO_2	Fe	SiO_2		
Low recovery....	52.30	17.90	56.50	11.60	50.50	20.70	30.0	32.4
High recovery...	52.30	15.50	54.50	12.70	37.70	39.65	88.1	91.4

Formation and Properties of Single Crystals of Synthetic Rutile

By CHARLES H. MOORE, JR.,* Member AIME

Introduction

In the study of the properties of rutile pigments it became apparent several years ago that certain physical and optical properties could not be determined on particles of pigmentary size. Since reflected light is the dominant type which reaches the eye from small particles, the true color of pure rutile was not known. Most rutile pigments are acicular in habit, elongated parallel to the "c" axis. It was considered important to know the nature and tone of light transmitted, for example, by a basal plate. Further, as shown by spectrophotometric curves, there is a very strong absorption of light of 4000 Å as measured on particles with random orientation. It would be interesting to see whether this absorption position varied with predetermined and selected orientations. Also, since the behavior of titanium pigments in a vehicle is important, a knowledge of the surface wettability of single crystals by various media should prove very fruitful. Finally, the optical properties of rutile are such that it should show, in a single pure crystal, greater fire and brilliance than the diamond. Mineralogists had long speculated that this would be the case and curiosity was strong to produce a single crystal large enough to cut.

In light of the above, a project was initiated for the formation of single crystals of rutile.

Method of Formation of Rutile Single Crystals

Fortunately, purification of TiO_2 produced from both titanium tetrachloride and titanyl sulphate solutions had reached the stage where spectrographically pure starting material could be made. Spectrographic analysis of the feed material used in the initial experiments is compared with the present feed material in Table 1. It is apparent that the present large scale purification is approximately as good as the original experimental, small scale purification.

Verneuil[1] produced synthetic ruby and corundum in 1904 by the well-known flame fusion process, which today bears his name. Except for mechanical improvements the present commercial production of synthetics by flame fusion is essentially unchanged from his initial method. His burner consisted of an inner orifice through which was fed the feed material and oxygen. Surrounding this was a larger outer orifice through which was fed hydrogen at lower pressure. This arrangement consists then of a lance of oxygen burning in an atmosphere of hydrogen. The proportions of hydrogen to oxygen usually used in corundum production is about 3:1, providing there is sufficient temperature to reach the 2050°C required for the fusion of alumina.

From work done in the Titanium Division Laboratories[2] and from the published work of Erlich[3] it was known that rutile appears to lose oxygen near its melting point and that it would not be possible to grow single crystals in a strongly reducing environment. Consequently, in order to determine whether single crystals could be grown at all, an Airco oxygen-acetylene torch with

Table 1 . . . Comparison of Original and Present Feed Material

	Original Feed Material	Present Feed Material
SiO_2	<0.02	0.04
Fe_2O_3	<0.001	<0.001
Al_2O_3	<0.001	0.001
Sb_2O_3	<0.002	<0.002
SnO_2	<0.001	<0.001
Mg	0.0005	0.0005
Cb	<0.005	<0.005
Cu	<0.0001	0.0001
Pb	<0.002	<0.002
Mn	<0.00005	<0.00005
W	<0.005	<0.01
V	<0.0005	<0.002
Ca	<0.01	
Cr	<0.0001	
Ba	0.001	

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TP 2617 H. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before July 30, 1949. Manuscript received Nov. 23, 1948.

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[1] References are at the end of the paper.

twin nozzles was set up as shown in Fig 1. A hopper, consisting of a 6 in. length of mullite furnace muffle 1½ in. in diameter was mounted vertically, with the bottom covered by a 100 mesh wire screen. A glass funnel leading to a copper tube was mounted below the hopper, with the end of the feed tube terminated an inch above the junction of the flames from the two tips. The feed material was vibrated through the screen by means of a small magnetic vibrator attached to the side of the hopper.

The tips of the torch were bent 45°, so that they faced each other. They were wound with 1/16 in. copper cooling coils and inserted through holes cut into a porous refractory brick. A hole was cut in the center to admit the feed material and the brick cemented to the torch with sodium silicate cement.

A preliminary furnace shell with an inside diameter of 2 in. was cast from the standard clay mix and the brick top with the torch inserted was rested upon it. In order to lower the fused material away from the flame, the furnace assembly was placed upon a scissors automobile mechanical jack, and the furnace lowered by hand. Small crystals, approximately 1 to 2 mm in diameter and 4 to 5 mm in length were grown with this apparatus. This showed that in spite of its tendency for very rapid crystallization rutile single crystals could be grown by the flame fusion method. It was obvious, however, that with a point fusion zone, no crystals of appreciable size could be grown and that any crystals produced would be so badly strained as to be useless. In consequence we designed a burner which would provide the minimum reducing environment and which would enlarge the area of constant temperature, to yield a broader fusion zone without a horizontal thermal gradient.

Where hydrogen and oxygen burn, an intense heat is created at the zone of mixing of the two gases. Turbulence of the gases broadens the zone but it is still a single reaction area. However, if an envelope of oxygen is made to surround the hydrogen to yield a three component, or three envelope, nozzle burning flame, there are two reaction zones with sufficient turbulence so that a constant temperature is maintained over the effective diameter of the nozzle. In addition, the outer oxygen envelope prevents excessive reduction of the TiO_2. A 0.5 in. nozzle diameter burner was built according to this design and single unfractured crystals of pure rutile, 1 in. long and 0.45 in. in diameter, were grown. The burner is virtually fool-proof, and the only critical factors are the relative space velocities of the three gas streams.

FIG 1—The first apparatus for attempted growth of single rutile crystals.

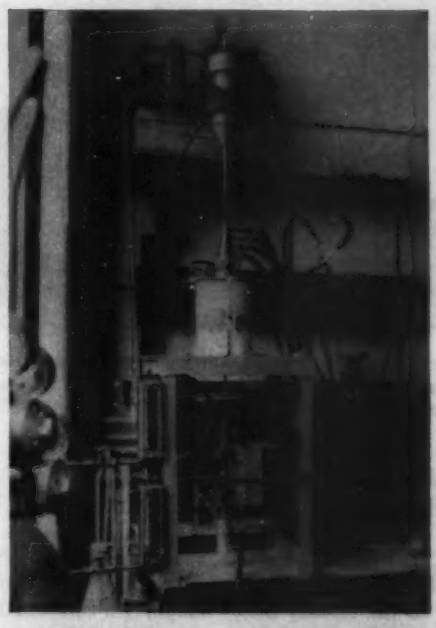

FIG 2—First burner made according to Titanium Laboratory design for rutile crystal growth.

In order to prevent periodic cooling of the flame by the showers of feed material produced by the customary tapping of the hopper, a very sensitive hopper was designed and a vibrator employed in place of the tapper commonly used on commercial burners. The constant flow of feed not only prevented the strain in the crystal which had formerly resulted from periodic cooling of the flame, but also eliminated the layer structure characteristic of Verneuil Process synthetic

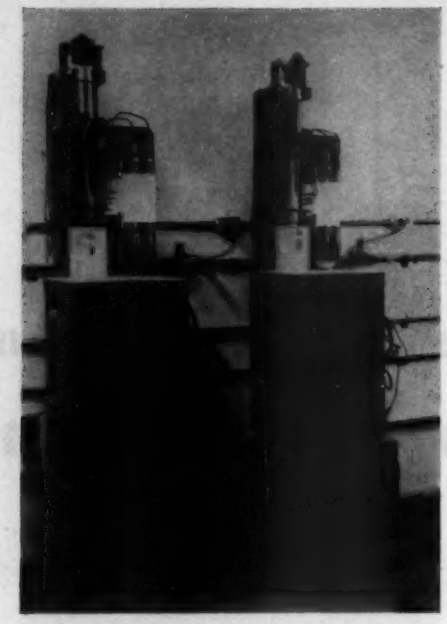

FIG 3—Improved burners and apparatus currently in operation for rutile single crystal growth.

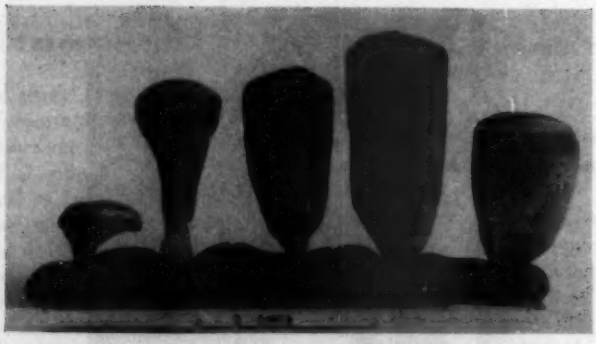

FIG 4—Rutile single crystals showing improvement in single crystals. Early growth is illustrated by the two boules at left and current production by the three boules at right.

crystals. This layer fusion had been deemed by Verneuil as an essential part of the flame fusion technique.

It was also felt that the hand lowering of the growing boule must contribute to strain in the crystal, since any operator must of necessity lower the boule a minimum of several millimeters if he is to keep up with the rate of boule growth and hold the fusion zone at a constant position in the flame. Therefore, a variable speed motor was attached to a worm gear and the pedestal lowered automatically. A provision was made also for a variable speed rotation of the pedestal in the event this was found to facilitate crystal growth. This original design burner and apparatus is illustrated in Fig 2.

The control of flow of oxygen and hydrogen to the burner is of the greatest importance. This was accomplished, in the 0.5 in. burner, by reduction valves on the tanks followed by low pressure diaphragm valves in the line; and which were followed in turn by needle valves in front of the rotometers. A simultaneous shut-off valve also was used. A filter of Spanish moss placed in the line removed excess moisture from the gases but is not considered essential when gases of normal purity are used. Pigtails connecting two tanks each of hydrogen and oxygen gave sufficient capacity of gas so that a 6 hr run could be made without shutting off, and without marked gas fluctuation.

The final burner design is essentially the same, except for engineering improvements, as the 0.5 in. nozzle burner. The present burners are 0.75 in. nozzle diameter and on which the hopper and lowering mechanism have been materially improved. These burners are connected to fixed lines which are fed from a series of 5 hydrogen and 5 oxygen tanks (see Fig 3).

Mechanism of Growth

A congruently melting compound which has a melting point within the range of the temperature of combustion of hydrogen and oxygen is quite simple to grow as a single crystal. The only absolutely essential features are accurate control of gases, a burner which will yield a fusion zone with a low horizontal thermal gradient, and a perfectly straight spindle under the pedestal. The control of particle size and shape is essential for a commercial operation, since the particles must be absorbed into the molten surface and melted completely as rapidly as possible.

Single crystals of rutile are grown in the following manner. The burner is cut on, the sintered alumina furnace (2½ in. id) is closed and the chamber allowed to heat for several minutes. The pedestal, consisting of any sufficiently refractory ceramic material, usually alumina or stabilized zirconia, is raised until it is just below the hot zone of the flame, which has a very steep vertical gradient. The vibrator is then turned on and the feed material drops on the pedestal. The temperature of the flame is held below the melting point of the compound by holding the hydrogen content of the flame low. A steep sintered cone is built up until the tip is approximately 1 mm across. The hydrogen flow rate is then increased and the tip melted. If frozen at this point the tip is found to consist of 3 to 5 crystals with various orientations. If not frozen, it is allowed to grow up into the hottest zone of the flame. It is essential at this stage that the flow of feed be very concentrated, since the "foot" must be kept narrow. This is necessary in order for one crystal to assume dominance and "freeze out" the others. After the foot has grown to a height of 4 to 6 mm, which takes about 5 min, the gas velocity is increased, the automatic lowering turned on and the width of the growing crystal is increased. This increase in velocity is continued until the desired width is obtained. The lowering rate is then synchronized with the rate

of crystal growth and the mechanism is allowed to run until a boule of requisite size is attained, when all gases are cut off simultaneously. After cooling in the furnace for 30 min the boule is removed. Although boules of large size can be grown, because of the increase in strain with size, the boules are restricted to approximately 100 carats (see Fig 4).

Growth from a seed crystal is much simpler since the very critical "foot" stage is eliminated. However, this is not considered a desirable technique for commercial production because of the manpower-time requirements. Seed are usually cemented to the pedestal by an alumina-clay cement. The crystallographic orientation obtainable by growth from a seed does not appear necessary with rutile. Crystals grown from cones have the "c" axis less than 20° off the vertical axis of the boule.

Characteristics of Single Rutile Crystals

Rutile single crystals grown by the above process are opaque black when removed from the furnace. However, because of the outer oxygen envelope, the rutile structure holds and there is no increment of Ti_2O_3 in the entire structure. A measurement of the unit cell of this black material is nearly identical with that of the fully oxidized rutile structure, both having a tetragonal unit cell. The colorless crystals show 4.5815 and 2.9509 Å, and the black shows 4.5820 and 2.9510 for the a and c directions respectively.* This black material is a strong semiconductor, and has a specific gravity of 4.268. It is possible to convert this material to colorless rutile by heating in a stream of oxygen. The oxygen addition to the crystal is directly proportional to the size of the crystal being reoxidized and the temperature used. The degree of reoxidation is readily checked by color changes, as the material goes from black through deep blue to light blue to green to colorless with a yellow tone. The specific gravity of the clear material has been found to vary between 4.29 and 4.39.† As will be shown, this variation in optically identical material was also noted in the dielectric measurements.

Table 2 shows hardnesses of the oxidation states of TiO_2 single crystals, measured by the diamond indentor, commonly known as the Knoop microindentor.

Table 2 . . . Hardness of TiO_2 Crystals by Knoop Microindentation Method

Sample	Orientation				Pyramid Face Cut, 45° to "c" Axis	Standard		
	Basal Plane Perpendicular "c" Axis		Prism Face Parallel "c" Axis					
	Parallel One Prism	Parallel Other Prism	In Direction of "c" Axis	Perpendicular to "c" Axis		Mineral	Mohs Hardness	Indentation No.
Perfectly orientated colorless cube	898	898	805*	840		corundum	9	±1,800
Yellowish over-oxidized crystal					940	topaz	8	±1,200
Light blue partially reduced cube	750	850	831*	835	890	quartz	7	790 710
Dark blue cube	766	840	859*	885		orthoclase	6	600–690
Black	940	950	1,000*	767				

* Indicates erroneous readings caused by excessive brittleness.

Table 3 . . . Change of Properties Upon Reduction

No. of Reduction Exposure	Crystallographic Direction	Oriented Cube		Rectangular with Long Direction Parallel to "c" Axis		Rectangular with Long Direction Parallel to "c" Axis	
		Dielectric Constant	Resistivity (ohm cm)	Dielectric Constant	Resistivity (ohm cm)	Dielectric Constant	Resistivity (ohm cm)
0	a	155	3.14×10^9	149	15.0×10^{13}	187	3.79×10^{13}
	a	275	4.5×10^{10}				
	c	83.5	1.55×10^{13}				
1.	a	4.9×10^4	1.7×10^4	4.68×10^4	1.13×10^4	2.34×10^4	4.73×10^4
	a	3.88×10^4	4.82×10^4				
	c	3.16×10^4	6.33×10^3				
2.	a	5.7×10^4	3.82×10^4	5.35×10^4	2.35×15^4	3.01×10^4	7.12×10^4
	a		1.92×10^4				
			3.0×10^3				
3.	a	4.58×10^4	4.2×10^4	4.15×10^4	3.39×10^4	2.22×10^4	9.5×10^4
	a	4.58×10^4					
	c		1.55×10^4				
4.	a	4.42×10^4	1.275×10^4	2.95×10^4	2.82×10^4	1.89×10^4	2.37×10^4
	a	4.24×10^4	2.24×10^4				
	c	3.64×10^4	1.55×10^4				
5.	a	4.31×10^4	2.55×10^4	3.17×10^4	2.62×10^4	1.62×10^4	1.66×10^4
	a	4.37×10^4	4.82×10^4				
	c	4.8×10^4	2.64×10^4				
6.	a	4.2×10^4	2.12×10^4	3.22×10^4	2.82×10^4	1.79×10^4	9.47×10^4
	a	3.53×10^4	7.06×10^4				
	c	4.7×10^4	2.18×10^4				
7.	a	4.25×10^4		3.27×10^4	9.42×10^3	1.81×10^4	
	a	3.88×10^4	8.02×10^4				9.47×10^4
	c	5.67×10^4	4.97×10^4				
8.	All samples indicate strong decay of dielectric constant during measurement.						
9.	Cannot measure dielectric constant—resistivity too low.						

The synthetic material as measured by the Knoop microindentation method is universally harder than natural rutile, in that the latter is reported having a hardness of 6 to 6½ Mohs scale, while the synthetic is as hard or harder than quartz in all directions. However, hardness measured in the direction of the "c" axis on the prism face exhibited a "butterfly" fracture effect around the indentation. This indicates a relief of stress in a sharp angle cone, with the length of the indentation correspondingly shortened. This is brought about by an excessive brittleness in this direction, making the indentation measurements in the direction of the "c" axis unreliable. Such a phenomenon would be expected to occur if there exists an atom deficiency in the planes parallel to "c" leaving many broken bonds, or if the atoms are slightly displaced along these planes, creating a state of strain.

Table 3 shows the dielectric measurements of single rutile crystals. These data show a marked change in dielectric properties, corresponding to the degree of "reduction" of the crystals. The measurements were made by the Crystal Section of the Naval Research Laboratory,[4] using a General Radio 716 capacity bridge in conjunction with a simple sample holder. All samples were colorless material reduced by H_2 at 600°C for the various time intervals.

* Measurement made by W. F. Sullivan of Titanium Division, Research Laboratory.
† Measurements were made by the volume displacement method.

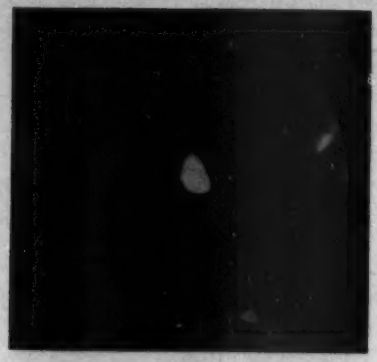

FIG 5—Stone cut with table face perpendicular to "c" axis. Culet appears without effect of birefringence. (X 10 magn.)

FIG 6—Stone cut with table face at 45° to "c" axis. Birefrigence causes culet to appear at two overlapping openings. (X 12½ magn.)

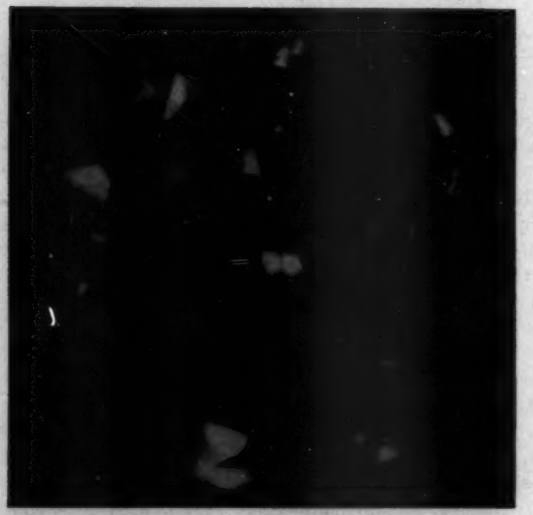

FIG 7—Stone cut with table face parallel to "c" axis. Birefrigence now causes culet to appear as two distinct openings. (X 12½ magn.)

The samples were held at 5 min intervals for the first five steps, 10 min intervals for the next two steps and 30 min intervals for the last three steps (with the latter held at 700 to 800°C). It was observed on making the reduction tests on the oriented cube, that the cube was colored a faint blue in the region of the basal faces after 10 min while still colorless in the central portions. As reduction time increased the blue zone moved into the crystal from both basal planes. This supplements the evidence shown by the hardness tests that the greatest loss of the oxygen upon reduction is in the direction of the "c" axis. Since TiO_2 is tetragonal, one would expect, if the change from colorless to black is caused by loss of planes of oxygen atoms, that this loss would be in a preferred direction. This is indicated by the evidence.

In light of the physical data, certain facts are noted concerning the single crystal rutile structure:

1. It is possible to vary these crystals from clear to opaque black without apparently affecting the volume or dimensions of the tetragonal unit cell.

2. There is a distinct difference in bonding between the planes of atoms parallel to the "c" axis and those in other directions.

3. Differences in hardness, brittleness, specific gravity and dielectric constant occur in crystals which are identical in color.

4. A given crystal can be taken through a complete cycle from colorless to a black semiconductor, apparently identical with the original furnace product, and back again to colorless, at least four times without materially affecting its properties. After six to eight cycles, the crystal becomes embrittled. The ability of the crystals to reduce and oxidize without structural change indicate an elasticity of structure not generally recognized.

The concept that removal of oxygen atoms brings about a darkening of the color upon reduction, to yield an increasing increment of Ti^{+3} may be correct, but it is difficult to apply to oxidation. A structure for the clear rutile analogous to FeO and FeS_2 should be considered as a possibility. These compounds exist as stable lattices with nonstoichiometric proportions of constituents. The strong covalent bonds necessary for such a cation deficient structure to exist have been shown, by dielectric studies, to occur in rutile.[5]

Rutile pigments show a strong ab-

FIG 8—Two half boules, the lower one dark blue and the upper one completely oxidized, with stones cut from each type. Smallest stones are 1.5 carats.

sorption of light at 4000Å to yield a higher increment of longer wavelengths in the transmitted light which results in a yellow tone. Single crystals of rutile are opaque to all wavelengths of light between 2100 and 3200Å and transparent from 4100 to 6800Å at 77°K.[6]

Synthetic Rutile As Gem Stones

The indices of refraction of synthetic rutile, measured by a three circle goniometer on a 20° prism cut parallel to the "*c*" axis showed 2.605 ± 0.004 for the ordinary ray and 2.901 ± 0.004 for the extraordinary ray. Sodium light was used. Measurements made by B. W. Anderson and C. J. Payne[7] on a cut rutile gem show very similar results. These values are within the range of the refractive indices of natural rutile. The refractive index, the marked dispersion, and the extremely high birefringence yields a brilliance and fire unrivalled by any other gem. The birefringe gives rise to a marked phenomena in the cut stones as illustrated by Fig 5 to 7. These photographs are taken of cut and faceted stones cut perpendicular to "*c*" axis, 45° to "*c*" axis and parallel to "*c*" axis, respectively. The well-known illustration of the double refraction for different optical orientations is clearly shown by the single, overlapping double and apparently separate culets at the base of the stones.

Even though the stone measured by Anderson and Payne was not cut to yield maximum values, the dispersion of the lowest index of refraction (0.2851) is six times that of a diamond. This dispersion of the extraordinary ray is even higher. This, of course, accounts for the tremendous "fire" of the material.

The standard brilliant or diamond cut is satisfactory for rutile. Since this type of cutting is designed to give total reflection of light by the facets of material with a critical angle of 24° (diamond) it will, of course, also serve for material with a critical angle of a maximum of 18° (rutile). However, if desired, the 6° advantage rutile has over diamond could be utilized to allow a wider bottom internal angle (100° is critical for diamond) yielding a larger diameter stone for its weight. Fig 8 shows boules and cut stones of synthetic rutile.

Since the colorless rutile is not a duplication or imitation of a natural stone as are the other synthetics, it is probably the first truly new gem since the advent of modern jewelry. It is the first material whose optical properties are such that it is superior to diamond in both brilliance and fire. Its future in this industry at least seems assured.

Acknowledgment

The writer wishes to express his appreciation to Dr. R. Dahlstrom, Technical Director, Titanium Division, National Lead Company, who made this investigation possible; to Drs. Paul Egli and S. Zerfoss and the staff of the Crystal Section of the Naval Research Laboratory for the electrical measurements; and to H. F. McMurdie of the National Bureau of Standards for refractive indices determinations. He also wishes to thank Dr. B. Levin, Chief of the Microoptical Section, Squier Signal Corps Laboratory, Fort Monmouth, N. J., for use of their equipment for the hardness testing.

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Coal Washing in Washington, Oregon, and Alaska*

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Coal washing assumed an important role in the mining industry of the Pacific Northwest long before washing practice became firmly established in the Appalachian field. A Scaife washer was operated in the state of Washington in 1887; and by 1927, the first year for which complete statistics were compiled, nearly a third of the state's coal production was washed, in comparison with only about 5 pct for the country as a whole. Mechanical preparation was adopted in Alaska in 1922 in a plant constructed by the United States Navy at Chickaloon to provide bunker coal for naval forces operating in northern waters.

Many of the coal beds mined in Washington and Alaska contained more impurities than those mined elsewhere, and this circumstance contributed to the early interest in mechanical cleaning. A much more important factor, however, was the inclination of the coal beds. With steeply pitching beds, hand sorting by the miner at the face is impossible; consequently, all material must be loaded and dealt with on the surface, just as is now proving the case with mechanical mining of flat beds. Thus "full-seam" mining afforded the same stimulus to the early development of coal washing in the Northwest that it is providing under mechanization in the rest of the country today.

Object and Scope

This report is intended to provide a general summary of the status of coal-washing practice in Washington, Oregon, and Alaska—an area in which some of the coals mined are more difficult to wash than those mined elsewhere in the country. Detailed flowsheets of plants, statistical data on production, and cost figures have been omitted, largely for the sake of brevity but also because such information is principally of only local interest. On the other hand, considerable washability data have been included because they are required to indicate the type of washing problems encountered in this area.

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[1] References are at the end of the paper.

Washington

At present Washington leads all other states in the percentage of its total production that is cleaned mechanically, 82.1 pct in comparison with the national average of 25.6 pct in 1945.[1] The washing problems encountered in this state are highly variable because the mountain-building forces that created the Cascade Range caused such intense folding and faulting of the beds in some fields that the rank of the coal was increased to anthracitic, while in other fields more distant from the mountains the beds were relatively undisturbed, and the rank of the coal ranges down through subbituminous to lignitic.

The complicated and burdensome schedule of sizes prepared in most coal fields has never been required by the Washington market. With but few exceptions, lump is prepared on a round-hole screen of about 3 in. size, egg is about 3 to 1⅝ or 1½ in., and nut is 1⅝ or 1½ in. to either 1 or ¾ in. Some mines screen no finer than ¾ in. and market a slack coal of that top size, but generally the coal is screened at about ¼ in. to give a stoker coal of ¾ or 1 to ¼ in., and a ¼ in. to 0 "buckwheat." During recent years the market for lump coal has largely disappeared, and lump is crushed to supplement the pro-

duction of stoker coal at most mines. In fact, at many operations shipments of lump coal rarely exceed 5 pct of the output.

ROSLYN-CLE ELUM FIELD

The areas of the Roslyn-Cle Elum field worked up until recent years are characterized by beds dipping only 10 to 15°, and the principal bed, the Roslyn, is relatively clean. Thus mechanical cleaning was not adopted in this field until 1935, when mining on steeper pitches and the introduction of mechanical loading in the flatter areas had progressed to a point where the amount of impurity hoisted with the coal was excessive. In that year cleaning plants were built by the two principal companies operating in the field, the Northwestern Improvement Co., coal-producing subsidiary of the Northern Pacific Railway, and the Roslyn-Cascade Coal Co.

Coal from the Roslyn bed is high-volatile A bituminous in rank. Considerable shale, clay, and a moderate amount of bony material contaminate the run-of-mine product, but it contains less than 1 pct of sulphur. Roslyn-bed coal, unlike many of the coals from western Washington fields, is only moderately difficult to wash. The quantity of intermediate-density material, on which the difficulty of washing chiefly depends, is, for example, less than that in the Thick Freeport bed of Pennsylvania but more than that in the No. 6 bed of southern Illinois. A specific-gravity analysis of 3 in. to 0 raw coal from the Roslyn bed is shown in Table 1.

Table 1 . . . Specific-gravity Analysis of Raw 3 In. Slack from Roslyn Bed

Specific Gravity	Weight, Per Cent	Ash,* Per Cent	Cumulative	
			Weight, Per Cent	Ash,* Per Cent
Under 1.30	46.5	6.6	46.5	6.6
1.30 to 1.40	38.1	12.4	84.6	9.2
1.40 to 1.60	7.2	27.1	91.8	10.6
1.60 to 1.80	1.8	47.4	93.6	11.3
Over 1.80	6.4	75.7	100.0	15.4

* Moisture-free basis.

The central cleaning plant of the Northwestern Improvement Co. was built to serve three mines, Roslyn Nos. 3, 5, and 9; it has a capacity of 200 tons per hour. At the individual mine tipples, the raw coal is screened at 3 in. round-hole size on shakers, the oversize hand-picked, and the clean lump crushed and added to the 3 in. screenings to constitute the washery feed. At the cleaning plant the raw coal is prescreened on a vibrator to give 3 to 1½ in. egg, 1½ to ¾ in. nut, and ¾ in. to 0 slack. Each of these sizes is washed separately in Vissac, pulsator-type jigs of 60 tph capacity; separate single jigs serve for the egg and nut sizes, and the ¾ in. slack is split between two units. The washed product from two slack jigs is screened on a vibrator dressed with ¼ in. cloth and the undersize rewashed on a group of six Deister-Overstrom tables.

A refuse retreatment circuit was installed in the plant several years ago. Refuse from the egg and nut jigs is crushed to 1 in. top size and, together with the refuse from the slack jigs, is rewashed in a small Elmore jig. Washed coal from the Elmore jig is crushed to pass ¼ in. and joins the ¼ in. to 0 material screened from the washed slack to form the feed for the wet tables. Final washed coal from the tables is recovered in a drag-type settling tank and dewatered in a centrifugal drier; the effluent of the centrifugal drier is returned to the fine-coal circuit by way of the usual conical settling tank. The slack coal is further dewatered in a Vissac heat drier during the winter months.

After washing, the egg and nut sizes are rescreened on shakers to remove degradation material. Egg, nut, 1½ in. slack, and ¾ in. slack are the sizes regularly shipped, and much of the coal goes for railway use. The plant is operated under laboratory control, with samples for ash analysis collected from each car of coal shipped.

The second washing plant in this field, operated by the Roslyn-Cascade Coal Co., treats coal from the No. 6 bed. As shown by the specific-gravity analysis in Table 2, this coal is somewhat more bony than that from the Roslyn bed.

Table 2 . . . Specific-gravity Analysis of Raw 3 In. Slack from No. 6 Bed

Specific Gravity	Weight, Per Cent	Ash,* Per Cent	Cumulative	
			Weight, Per Cent	Ash,* Per Cent
Under 1.35	60.3	9.5	60.3	9.5
1.35 to 1.50	23.4	20.7	83.7	12.6
1.50 to 1.70	5.7	37.4	89.4	14.2
Over 1.70	10.6	74.7	100.0	20.6

* Moisture-free basis.

The run-of-mine product is screened at 3 in., the oversize lump hand-picked, and most of the clean lump crushed in a pick-breaker to join the 3 in. screenings constituting the washery feed. A 5-cell, 2-compartment, Baum-type jig of 80 tph capacity is used for cleaning 3 to ¼ in. coal, and a Stump air-flow unit is used for the ¼ in. to 0 slack After washing, the coal is sized on shakers to produce 3 to 1½ in. egg, 1½ to ¾ in. nut, and ¾ in. slack, the sizes normally shipped. A Ruggles-Cole heat drier is used for reducing moisture in the ¾ in. slack during the winter months, and this product can be oil-treated for dustproofing if desired.

WESTERN WASHINGTON FIELDS

About 30 mines, most of them small truck operations, are active in western Washington at present, and over two-thirds of this number have washeries. The flowsheets employed at the small mines are almost identical; lump is prepared at 3 in. on a shaker, and the 3 in. slack is washed in a Forrester jig. In fact, the Forrester jig is standard equipment at small mines in the state. A single-cell, piston-type unit, the Forrester relies on manual control of refuse removal and hence is "only as good as the operator."

Largest of the western Washington plants, the washery of the Bellingham Coal Mines, located at Bellingham, has a capacity of about 1000 tons per day, using Forrester jigs for washing 4 to 1½ in. coal, a Faust jig for nut coal, and wet tables for material finer than 5/16 in.

A 5-cell, 2-compartment, Baum-type jig installed several years ago at the Black Diamond washery of the Palmer Coking Coal Co. is employed for the 3 in. slack from several small mines operated by the company. Coal from a strip pit on the No. 12 or Fulton bed, a specific-gravity analysis of which is shown in Table 3, is particularly difficult to wash. This difficulty is illustrated in Fig 1 which compares the Fulton coal with the Roslyn and No. 6 bed coals previously described. It contains nearly 25 pct of impurity heavier than 1.70 sp gr and a large proportion of intermediate-density material. Moreover, much of the impurity is clay of such a plastic character that small particles adhere tenaciously to pieces of clean coal and can be removed only by vigorous sprays on the washed-coal sizing screens.

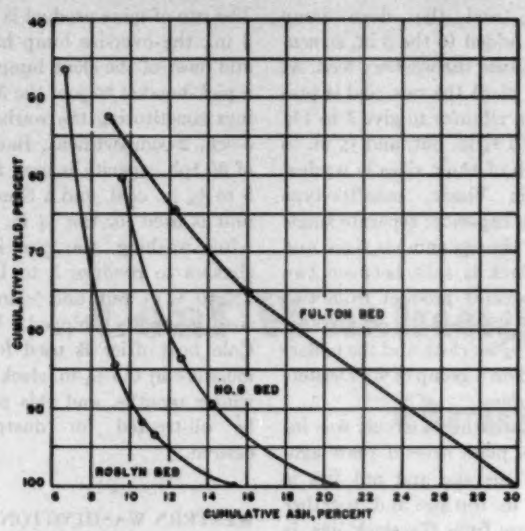

FIG 1—Yield-ash curves for Roslyn, No. 6, and Fulton beds.

Table 3 . . . Specific-gravity Analysis of Raw Slack from Fulton Bed

Specific Gravity	Weight, Per Cent	Ash,[a] Per Cent	Cumulative Weight, Per Cent	Cumulative Ash,[a] Per Cent
Under 1.40	52.4	8.9	52.4	8.9
1.40 to 1.50	12.1	26.5	64.5	12.2
1.50 to 1.70	10.6	39.8	75.1	16.1
Over 1.70	24.9	72.9	100.0	30.2

[a] Moisture-free basis.

Table 4 . . . Specific-gravity Analysis of 3½ In. to 0 Raw Slack from McKay Bed

Specific Gravity	Weight, Per Cent	Ash,[a] Per Cent	Cumulative Weight, Per Cent	Cumulative Ash,[a] Per Cent
Under 1.30	60.1	3.0	60.1	3.0
1.30 to 1.40	14.0	8.8	74.1	4.1
1.40 to 1.50	4.7	21.4	78.8	5.1
1.50 to 1.70	5.1	38.8	83.9	7.2
Over 1.70	16.1	71.6	100.0	17.5

[a] Moisture-free basis.

The McKay bed, best known of the western Washington coals, currently is mined at Ravensdale by the Northwestern Improvement Co. and at Cumberland by the Olson Fuel Co. A coal of subbituminous A rank, the McKay has an inherent ash content of less than 3 pct and thus is by far the cleanest coal in the state. The specific-gravity analysis of raw 3½ in. slack shown in Table 4 demonstrates, however, that with the mining conditions prevailing at Ravensdale a large quantity of impurity contaminates the raw coal.

A 2-compartment Elmore jig, recently rebuilt as a steel box, handles 3½ in. slack at Ravensdale. The washed coal is sized on vibrating screens to produce the conventional egg, nut, stoker, and buckwheat sizes. At this mine, as at many other operations in the state, the sludge or slurry overflowing the washed-coal settling tank is flumed to settling ponds from which it is recovered ultimately by scraper.

Pierce County contains the only reserves of coking coal in the state, and coals from the Wilkeson-Carbonado-Fairfax district of this county were used for coke manufacture from 1880 to 1936, when the demand for coke would no longer support continued production. With the advent of the late war, however, the demand for coke was renewed, and a mine at Wilkeson was developed by the Wilkeson Products Co. to supply coal for a byproduct-coke plant at Tacoma. The washery constructed for this mine is of interest principally because of the elaborate provision made for rewashing the coal. As shown by the specific-gravity analyses in Table 5, the beds mined at Wilkeson contain an excessive quantity of heavy impurity as well as considerable bony material. Fig 2 compares these beds by means of yield-ash curves.

To deal with material of this character, the raw coal was first put through a Bradford breaker to eliminate the coarse rock and break the remainder of the material to pass 1 in. The 1-in. slack was then washed in a Vissac jig set to produce a clean refuse and a washed product still containing some impurity. This washed coal was passed over a 1 mm wedge-wire screen, the oversize was rewashed in a second Vissac jig, and the undersize was rewashed on a battery of 8 Deister-Overstrom tables.

The washed product from the second Vissac jig was screened at 4 mm, the oversize material constituting finished clean coal and the undersize material being added to the table feed. Refuse from the second jig, in reality a middling product, could be either marketed as a high-ash steam coal or crushed to

Table 5 . . . Specific-gravity Analyses of Raw Wilkeson 1 In. Slacks

Bed	Specific Gravity	Weight, Per Cent	Ash,[a] Per Cent	Cumulative Weight, Per Cent	Cumulative Ash,[a] Per Cent
No. 1, west dip	Under 1.40	35.8	9.1	35.8	9.1
	1.40 to 1.50	13.2	23.4	49.0	13.0
	1.50 to 1.70	14.0	37.9	63.0	18.5
	Over 1.70	37.0	65.3	100.0	35.8
No. 2, west dip	Under 1.40	56.1	9.7	56.1	9.7
	1.40 to 1.50	13.0	24.3	69.1	12.4
	1.50 to 1.70	10.4	37.0	79.5	15.7
	Over 1.70	20.5	69.9	100.0	26.8
No. 3, west dip[b]	Under 1.40	47.8	9.1	47.8	9.1
	1.40 to 1.50	10.8	23.8	58.6	11.8
	1.50 to 1.70	10.8	37.8	69.4	15.9
	Over 1.70	30.6	70.6	100.0	32.6
No. 5, east dip	Under 1.40	78.5	8.1	78.5	8.1
	1.40 to 1.50	7.2	23.0	85.7	91.4
	1.50 to 1.70	5.4	36.3	91.1	10.9
	Over 1.70	8.9	65.1	100.0	15.8

[a] Moisture-free basis.
[b] ¼-in. slack.

pass ¼ in. and added to the table feed. Most of the final clean coal was produced on the tables.

This elaborate schedule of retreatment was required to produce a washed coal of 11 to 12 pct ash, corresponding to a separation at a little less than 1.50 sp gr, and demonstrates the type of washing involved if a very difficult coal must be washed at low specific gravity. The plant is not now operating and currently is offered for sale by the War Assets Administration.

Alaska

Coal production in Alaska during recent years has been divided about equally between the subbituminous coals of the Nenana field and the bituminous coals of the Matanuska Valley. The subbituminous coals, mined at Suntrana, are clean enough to be prepared by screening and some hand picking. In contrast, the coals of the Matanuska Valley contain more impurity than most so-called "dirty" coals mined in the United States and consequently must be washed to render them suitable for even the Alaska market. The proportion of washery feed rejected as refuse in Alaska averaged 33 pct in 1945, a far higher figure than that recorded for any state in the country.[1]

In 1943 the Alaska railroad built a washery at its Eska mine to prepare coal for railroad use. All coal was crushed to pass a 2½ in. bar screen and washed in a 3-cell, 1-compartment, Baum-type jig at the rate of 60 tph. Washed coal was sized into 1 in. nut and 1 in. slack on a vibrator. During the winter months coal finer than ⅜ in. was dewatered in a centrifugal drier. Detailed performance tests of this plant were made by the Bureau of Mines to determine the relationship between quality and yield of washed coal.[2] Since so little information is available on the operation of washers treating unusually dirty coal, some of the data from these tests are summarized here to illustrate the type of performance obtained.

Table 6, giving the specific-gravity analyses of the jig feed by several size fractions, indicates that 40 pct of the raw coal is impurity heavier than 1.70 sp gr and about 25 pct occurs between 1.40 and 1.70 sp gr.

Actually, the concentration of impurity in the material coarser than 1-in. is so great that this size contains

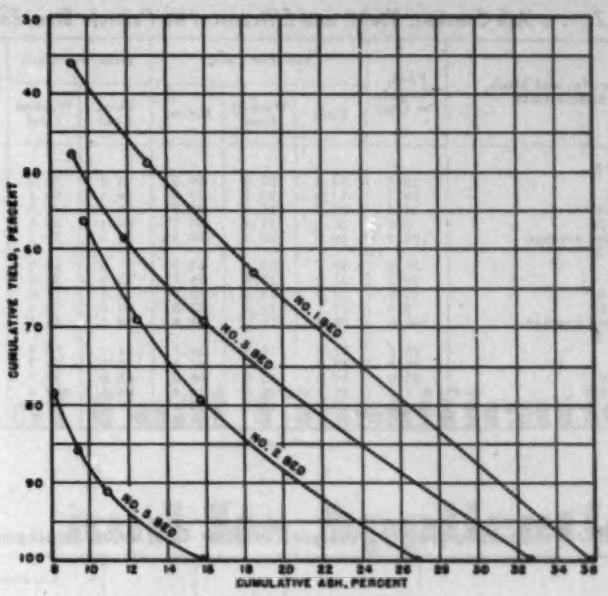

FIG 2—Yield-ash curves for Wilkeson coal beds.

little more coal than the refuse discarded at some washeries. Table 7 gives the ash contents, yields, and efficiencies obtained in a series of three tests in which the jig was operated to give a washed coal of increasingly lower ash content.

Efficiency, as the term is used in Table 7, is the ratio of the yield of washed coal to the yield of float coal of the same ash content present in the feed, expressed as a percentage. The efficiency of washing fell from 96 pct in Test 1, in which a washed coal of 20.4 pct ash was obtained, to 89 pct in Test 3, in which the washed coal contained 14.1 pct ash. These efficiencies are much lower than the values of 98 to 99 pct characterizing good washery performance, yet they

Table 6 . . . Specific-gravity Analyses of Various Sizes of Raw Coal, Eska Mine

Size	Specific Gravity	Weight, Per Cent	Ash,[a] Per Cent	Cumulative	
				Weight, Per Cent	Ash,[a] Per Cent
Over 1 in. Weight, 28.7 pct	Under 1.30	3.2	4.6	3.2	4.6
	1.30 to 1.40	6.1	14.6	9.3	11.2
	1.40 to 1.50	12.8	25.9	22.1	19.7
	1.50 to 1.70	18.7	38.8	40.8	28.5
	Over 1.70	59.2	72.2	100.0	54.4
1 to ⅜ in. Weight, 29.1 pct	Under 1.30	15.3	4.7	15.3	4.7
	1.30 to 1.40	13.0	14.1	28.3	9.0
	1.40 to 1.50	13.8	26.5	42.1	14.7
	1.50 to 1.70	15.3	40.1	57.4	21.5
	Over 1.70	42.6	72.9	100.0	43.4
⅜ in. to 20 mesh Weight, 35.4 pct	Under 1.30	35.9	3.6	35.9	3.6
	1.30 to 1.40	17.7	12.7	53.6	6.6
	1.40 to 1.50	9.3	25.3	62.9	9.4
	1.50 to 1.70	11.5	39.4	74.4	14.0
	Over 1.70	25.6	71.2	100.0	28.7
Under 20 mesh Weight, 6.8 pct	Under 1.30	37.4	2.9	37.4	2.9
	1.30 to 1.40	17.6	9.9	55.0	5.1
	1.40 to 1.50	6.7	24.1	61.7	7.2
	1.50 to 1.70	8.4	36.2	70.1	10.7
	Over 1.70	29.9	69.7	100.0	28.3
Composite, all sizes	Under 1.30	20.7	3.8	20.7	3.8
	1.30 to 1.40	13.0	13.1	33.7	7.4
	1.40 to 1.50	11.4	25.9	45.1	12.1
	1.50 to 1.70	14.5	39.3	59.6	18.7
	Over 1.70	40.4	72.1	100.0	40.3

[a] Moisture-free basis.

Table 7 . . . Ash Content, Yield, and Efficiency, by Particle Size, Eska Mine

Size, In. and Mesh, Square Hole	Feed, Weight, Per Cent	Ash, Per Cent*			Yield, Per Cent		Efficiency, Per Cent
		Feed	Washed Coal	Refuse	Float Coal	Washed Coal	
Test No. 1							
Over 1	28.7	54.4	25.9	69.1	35.2	34.2	97.2
1 to ¾	29.1	43.4	23.1	72.8	60.4	59.1	97.8
¾ to 20	35.4	28.7	16.4	65.8	79.3	75.3	95.0
Under 20	6.8	28.3	21.4	55.7	89.4	80.0	89.5
Weighted average	100.0	40.3	20.4	69.0	61.8	59.1	96.1
Test No. 2							
Over 1	26.9	53.7	20.4	64.9	27.6	25.1	90.9
1 to ¾	29.5	41.2	18.4	66.7	55.2	52.8	95.7
¾ to 20	35.6	28.0	14.2	62.8	76.2	71.6	94.0
Under 20	8.0	27.1	19.9	53.5	89.0	78.4	88.1
Weighted average	100.0	38.8	16.9	64.6	58.0	54.1	93.2
Test No. 3							
Over 1	28.6	51.5	16.9	59.3	21.6	18.4	85.2
1 to ¾	29.0	43.3	15.6	61.1	42.6	39.3	92.3
¾ to 20	35.0	30.6	12.0	58.0	65.8	59.7	90.7
Under 20	7.4	26.0	16.6	48.4	84.5	69.8	82.6
Weighted average	100.0	39.9	14.1	59.1	47.8	42.7	89.0

* Moisture-free basis.

Table 8 . . . Specific-gravity Analyses of Raw Coal from Southport Mine

Size, In. and Mesh	Specific Gravity	Weight, Per Cent	Ash,* Per Cent	Cumulative	
				Weight, Per Cent	Ash,* Per Cent
2 to 1½ in.	Under 1.40	73.2	9.8	73.2	9.8
Weight, 10.9 pct	1.40 to 1.50	10.6	22.9	83.8	11.5
Ash, 22.4 pct*	1.50 to 1.70	2.8	39.3	86.6	12.4
	Over 1.70	13.4	87.8	100.0	22.4
1½ to ¾ in.	Under 1.40	72.4	9.8	72.4	9.8
Weight, 21.3 pct	1.40 to 1.50	11.2	22.1	83.7	11.4
Ash, 22.1 pct*	1.50 to 1.70	3.4	40.3	87.1	12.6
	Over 1.70	13.0	85.9	100.0	22.1
¾ to ¾ in.	Under 1.40	72.4	9.4	75.4	9.4
Weight, 23.7 pct	1.40 to 1.50	9.9	21.2	85.3	10.8
Ash, 20.1 pct*	1.50 to 1.70	3.7	39.0	89.0	11.9
	Over 1.70	11.0	86.3	100.0	20.1
¾ in. to 20 mesh	Under 1.40	72.4	8.2	72.4	8.2
Weight, 36.2 pct	1.40 to 1.50	10.1	18.4	82.6	9.4
Ash, 19.3 pct*	1.50 to 1.70	3.6	33.6	86.2	10.5
	Over 1.70	13.9	74.0	100.0	19.3
Under 20 mesh	Under 1.40	49.0	8.0	49.0	8.0
Weight, 7.9 pct	1.40 to 1.50	11.0	18.4	60.0	9.9
Ash, 34.7 pct*	1.50 to 1.70	7.1	32.9	67.1	12.3
	Over 1.70	32.9	80.3	100.0	34.7
Composite	Under 1.40	71.3	9.0	71.3	9.0
Weight, 100.0 pct	1.40 to 1.50	10.4	20.4	81.7	10.5
Ash, 21.6 pct*	1.50 to 1.70	3.8	36.5	85.5	11.6
	Over 1.70	14.5	81.0	100.0	21.7

* Moisture-free basis.

do not represent particularly faulty jig operation. The percentage of coal in the refuse product was no higher than that obtained in normal jig operation, but the abnormal quantity of refuse rejected, ranging from 40.9 pct in Test 1 to 57.3 pct in Test 3, made this nominal percentage of coal constitute a serious washery loss.

With the conversion of the Alaska railroad from coal to oil, the Eska mine was closed and put in standby condition.

At the mine of the Evan Jones Coal Co., a few miles from Eska, the coal contains considerably less impurity but still constitutes a washing problem of more than usual difficulty. The egg, nut, and ¾ in. slack sizes are washed separately in a single Forrester-type jig of about 30 tph capacity. As might be expected, manually controlled equipment of this type is not particularly satisfactory for treating a difficult coal, even in the hands of an experienced, competent operator. Fluctuations in the quality of the washed coal occur, and the refuse product may contain more than the usual amount of coal.

Oregon

Coal occurs at a number of locations in western Oregon, but mining on a substantial scale has been limited to the Coos Bay field, on the southern coast of Oregon. Coos Bay coal was first washed at the Beaver Hill mine, which supplied fuel for the Southern Pacific Railroad in the early 1900's. The present Southport mine of the Coast Fuel Corp., started during the closing years of the recent war, also required a washing plant to remove the interbedded clay of the Beaver Hill bed. Located 5 miles south of Coos Bay, this plant incorporates provision for screening the run-of-mine product at about 4 in. size, hand-picking the lump, and optionally crushing the clean lump to join the screenings as feed for a Forrester-type jig. Washed coal is sized on a vibrating screen into the conventional egg, nut, and stoker sizes, with the material finer than ¼ in. screened from the stoker coal and flumed to a storage pond for possible future utilization.

Table 8 shows a specific-gravity analysis, by size fractions, of raw coal from this mine. The material represented was run-of-mine coal from which 5 pct of clean rock had been removed by hand picking. The remainder of the material was crushed to pass 2 in. square-hole. With less than 15 pct of impurity heavier than 1.70 sp gr and little material of intermediate density, the Southport coal is cleaner than most Washington or Alaska coals. A washed product of 12 to 13 pct ash, moisture-free basis, can be produced readily from this coal. The only difficulty encountered arises from the character of the clay that constitutes the principal impurity. This clay disintegrates readily in water and therefore tends to contaminate the finer sizes of coal, from which its removal is more difficult; the clay would also build up in a closed-circuit water system.

References

1. U. S. Bur. Mines Minerals Yearbook (1946).
2. M. R. Geer and H. F. Yancey: Washability Characteristics and Washing of Coals from the Matanuska Field of Alaska. U. S. Bur. Mines *R.I.* 3840 (1946).

A Study of Coal Classification and Its Application to the Coking Properties of Coal

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The fact that coal is a complex organic material and heterogeneous in composition has made its study extremely difficult, particularly in regard to obtaining a fundamental concept of the processes involved in the formation of coke. It has been stated[1] recently that coals are natural polymers and that in a complex reaction such as occurs during carbonization the differences between coals are quantitative and not qualitative. Such being the case, it would be expected, therefore, that a series of coals subjected to exactly the same conditions of coking would produce results that should be related quantitatively to the chemical analyses of the individual coals. In the past, however, this fact was not generally recognized and consequently many empirical test methods were devised for testing coals. The interpretation of the results obtained depended upon comparing a set of data with other data on coals of known characteristics. The testing procedure had to be conducted in precisely the same manner each time and not the least requirement in making the interpretation was experience. The test methods varied from those utilizing a gram of coal to the plant-scale tests in which one or more ovens were charged with the test coal. The aim had been principally to find some simple, inexpensive means by which the coke-making and other properties of coal could be predicted. It is not necessary to emphasize the importance of these endeavors, for anyone acquainted with the problems involved in the selection of coals for making coke is well aware of the time and expense involved in making a series of definitive tests.

Several attempts have been made to derive formulas for such predictions. The relations of both ultimate and proximate analyses to coke physical properties and byproduct yields have been studied. Lowry and co-workers[1] correlated statistically the U. S. Bureau of Mines-American Gas Association assay test results with coal analyses, using 90 coals and coal blends. Parry,[2] Spooner,[3] and Gabinskii and Krym[4] made similar correlations. Lowry concluded that there were definite relationships between the proximate analyses of the coals and the yields and properties of the products. Although this statistical study was made on test results from carbonizing coal in a cylindrical steel retort, the conclusions are important and provided one of the first steps in the simplification of this complex subject. Studies should be made of other references on this subject just cited.

Studies of the coking process reported in the literature frequently were confined to a small number of coals, and very few have included coals from various ranks. It would seem logical, however, that a program for studying factors involved in the formation of coke should be based on a number of coals selected from each of the various classes or types of coking coal. The possible use of the scheme of classifi-

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[1] References are at the end of the paper.

XUM

Table 1 . . . ASTM Standard Classification of Coals by Rank

Class	Group	Limits of Fixed Carbon or British Thermal Units, Mineral-matter-free Basis*	Requisite Physical Properties
I. Anthracitic	1. Meta-anthracite	Dry FC, 98 pct or more (dry VM, 2 pct or less)	Nonagglomerating
	2. Anthracite	Dry FC, 92 pct or more and less than 98 pct (dry VM, 8 pct or less and more than 2 pct)	
	3. Semianthracite	Dry FC, 86 pct or more and less than 92 pct (dry VM, 14 pct or less and more than 8 pct)	
II. Bituminous	1. Low-volatile bituminous coal	Dry FC, 78 pct or more and less than 86 pct (dry VM, 22 pct or less and more than 14 pct)	Either agglomerating or nonweathering
	2. Medium-volatile bituminous coal	Dry FC, 69 pct or more and less than 78 pct (dry VM, 31 pct or less and more than 22 pct)	
	3. High-volatile A bituminous coal	Dry FC, less than 69 pct (dry VM, more than 31 pct); and moist Btu 14,000 or more	
	4. High-volatile B bituminous coal	Moist Btu, 13,000 or more and less than 14,000	
	5. High-volatile C bituminous coal	Moist Btu, 11,000 or more and less than 13,000	
III. Subbituminous	1. Subbituminous A coal	Moist Btu, 11,000 or more and less than 13,000	Both weathering and nonagglomerating
	2. Subbituminous B coal	Moist Btu, 9,500 or more and less than 11,000	
	3. Subbituminous C coal	Moist Btu, 8,300 or more and less than 9,500	
IV. Lignitic	1. Lignite	Moist Btu, less than 8,300	Consolidated
	2. Brown coal	Moist Btu, less than 8,300	Unconsolidated

* FC, fixed carbon; VM, volatile matter; Btu, British thermal units.

cation of coal by rank based on chemical analyses as a tool in the study of the behavior of coking coal was emphasized by the authors in 1942.[5] It was said in part at that time, " . . . a good understanding of the principles involved in setting up the standard method of classification may lead to a better appreciation of the causes of changes in coke characteristics."

There is sufficient evidence obtained from past experience to indicate that characteristics of coke produced from a given coal are in some measure related to the rank of coal. Broadly speaking, the high-volatile C bituminous coals make very small fragile cokes if the coals coke at all; whereas the high-volatile A coals produce much better cokes. Although the low-volatile bituminous coals produce strong cokes, they cannot be carbonized unblended in byproduct coke ovens because they exert excessive pressures against the oven walls. While these generalities seem to be obvious, the question immediately arises as to whether or not small changes in rank produce quantitative differences in coking behavior.

The purpose of this study is to present a discussion of coal classification by rank with the introduction of a new scheme developed by the authors. Some preliminary results of correlating coking behavior with coal rank will be presented.

Classification of Coal

The fact that coal is a natural product derived from a wide variety of vegetable matter that has been altered by bacterial action, heat, pressure, and time gives rise to a continuous series in which there are no obvious boundaries between different types. A method by which coals could be classified was sought by a large number of investigators over a period of more than a hundred years. During that time a very large number of schemes were proposed. It would be impossible to present even a cursory account of the literature on classification. The reader is referred to an excellent review of the subject as written by Dr. H. J. Rose.[6]

The methods devised in the past may be divided into three categories: (1) methods based on proximate analyses, (2) methods based on ultimate analyses, and (3) methods based on both proximate and ultimate analyses. The heating value of the coal is included in one or another of the schemes of classification and this property is an important factor in the standard procedure for classification of coal by rank.

ASTM CLASSIFICATION BY RANK

Through the joint efforts of the American Standards Association, the American Society for Testing Materials, and the American Institute of Mining and Metallurgical Engineers, the present American standard for classification of coal was established about ten years ago. This development of a classification scheme by the unified efforts of a large group of scientists and technologists of acknowledged reputation was one of the outstanding accomplishments in coal technology. Despite the widespread interest and activity that extended over a period of more than ten years in which the method was developed, it is surprising that little attention has been paid to applying the method to coal carbonization.

The basis for the scheme of classification is the proximate analysis and the heating value of the coal. The fixed carbon of the coal is calculated to the dry, mineral-matter-free basis, and the heating value in Btu per pound is calculated to the moist, mineral-matter-free basis. Table 1 presents the details of the classification as designated by the ASTM. This scheme places all of the coking coals in the bituminous class. The ASTM Standards[7] should be consulted for the various details of the specifications. Definitions of agglomerating, nonagglomerating, weathering, and nonweathering are given because these factors are used to separate the bituminous class from the anthracitic on the one side, and

from the subbituminous on the other. Each of the groups merges into the adjoining one and the boundary lines between the groups are arbitrary limits established by agreement.

Fig 1 is a plot of more than 300 coal analyses of all ranks from meta-anthracite to lignite. This is a composite of the various charts shown in *R. I.* No. 3296 published by the U. S. Bureau of Mines with the exception that the scale of the abscissa is reversed. The coal analyses used in the construction of these classification charts and also including Fig 2, 4, 5, and 6 were selected by the U. S. Bureau of Mines from their various bulletins and technical papers on the analyses of United States coals. Although the method of classification that was finally adopted specifies the use of "bed moisture" for the correction of the heating value of coal to moist, mineral-matter-free basis, all analyses used in *R. I.* No. 3296 are on the "as-received" basis and in consequence the charts developed in this study of coal classification are also on the same basis.

More detailed attention must be given to the terms "mineral matter" and "moisture" in this discussion, since a thorough understanding of these terms is not only necessary in consideration of the standard classification but also of the modified classification to be described in this paper. Mineral matter is the ash material as it exists in the original coal. During the determination of ash, the water of hydration of the slate, and so on, is driven off, and the pyrite which is represented by the chemical formula FeS_2 is converted to Fe_2O_3. The ash that is weighed in the determination is consequently of different composition and weight than the original mineral matter. Parr and Wheeler[6] developed empirical formulas by which ash as weighed could be converted to the original mineral matter. Coal corrected to the mineral-matter-free basis was called "unit coal" by those authors, but the ASTM has preferred to designate this correction as "mineral-matter-free." The following are the formulas developed by Parr and Wheeler with some later amplification by these authors:

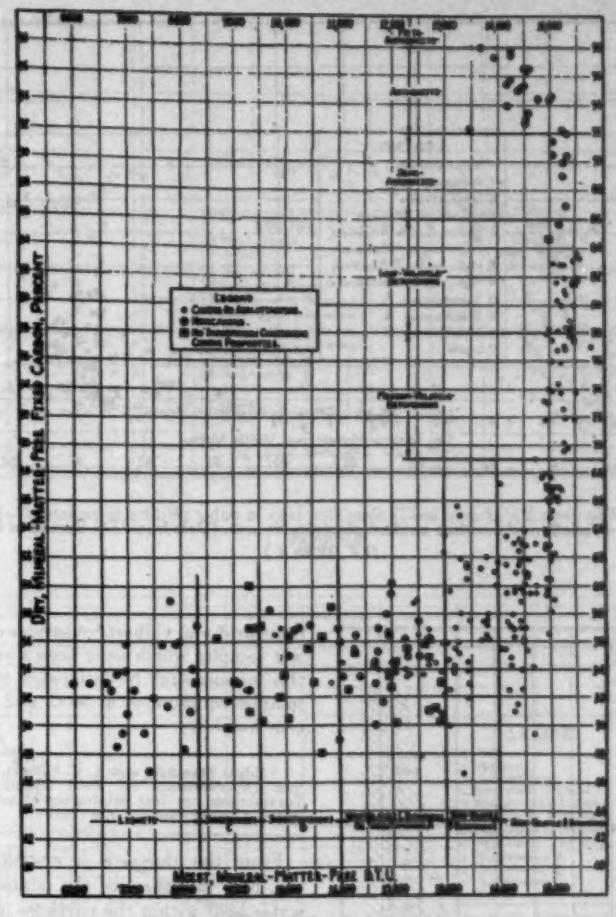

FIG 1—Classification of typical United States coals. (U.S. Bureau of Mines *R.I.* 3296.)

PARR FORMULAS

$$\text{Dry, Mm-free FC} = \frac{\text{FC} - 0.15\text{S}}{100 - (\text{M} + 1.08\text{A} + 0.55\text{S})} \times 100$$

$$\text{Dry, Mm-free VM} = 100 - \text{Dry, Mm-free FC}$$

$$\text{Moist, Mm-free Btu} = \frac{\text{Btu} - 50\text{S}}{100 - (1.08\text{A} + 0.55\text{S})} \times 100$$

Where: Mm = Mineral matter
FC = Fixed Carbon
VM = Volatile Matter
S = Sulphur
M = Moisture
A = Ash
Btu = British thermal units

Simplified formulas are given below called "Approximation Formulas." The Parr formulas are accepted in all cases where litigation is involved.

APPROXIMATION FORMULAS

$$\text{Dry, Mm-free FC} = \frac{\text{FC}}{100 - (\text{M} + 1.1\text{A} + 0.1\text{S})} \times 100$$

$$\text{Dry, Mm-free VM} = 100 - \text{Dry, Mm-free FC}$$

$$\text{Moist, Mm-free Btu} = \frac{\text{Btu}}{100 - (1.1\text{A} + 0.1\text{S})} \times 100$$

The moisture content of the coal used in the scheme of classification is the bed moisture. Because this factor is so important and possibly may be misunderstood, the following is a verbatim copy from the ASTM standard[7] in which the "moist" basis is specified:

(*d*) In case the coal is likely to be classified on the "moist" basis, that is, containing the natural bed moisture, the samples shall be taken at freshly exposed faces, which are free from visible surface moisture if possible. Samples of low-rank coals which appear dry at the time of collection frequently give off moisture which condenses on the inner surface of the sample containers, before they are opened for analysis. In the case of coals which were free from visible surface moisture when sampled, but which show moisture on the inner surface of the container when opened, both the container and the coal shall be weighed before and after

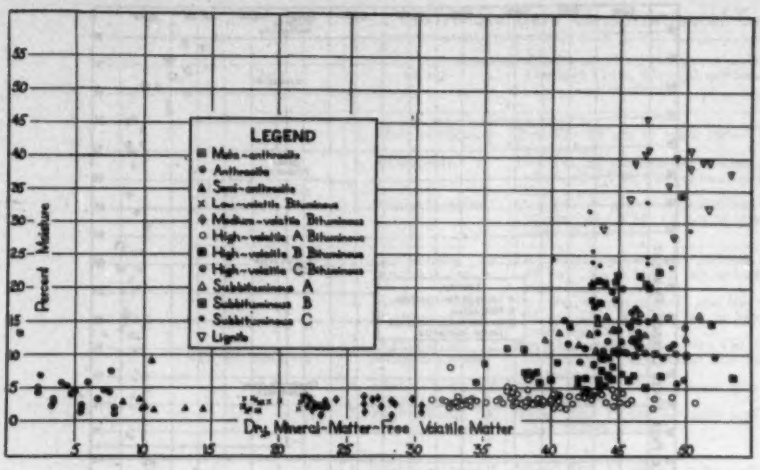

FIG 2—Variation of moisture content of coal with rank. (Data from U.S. Bureau of Mines *R.I.* 3296 R.)

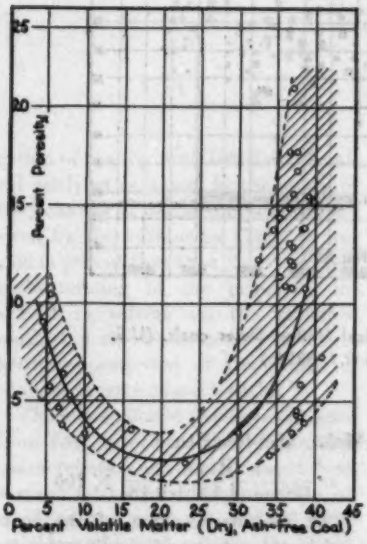

FIG 3—Variation of coal porosity with volatile matter. (Data from The Ultra-fine Structure of Coals and Coke p 48 BCURA 1944.)

air-drying and the total loss in weight shall be reported as air-drying loss.

(*e*) If it is impossible to sample the coal without including visible surface moisture, and the coal is likely to be classified on the "moist" basis, the sampler shall include the following statement in the description: "Sample contains surface moisture." Samples so marked shall not be used for classification on a moist basis unless brought to a standard condition of moisture equilibrium at 30 C. in a vacuum desiccator containing a saturated solution of potassium sulfate (97 per cent humidity) as suggested by Stansfield and Gilbart.[5] Analyses of such wet samples which have been treated in this manner shall be designated as "wet samples equilibrated at 30 C. and 97 per cent humidity.

[5] Edgar Stansfield and K. C. Gilbart, "Moisture Determination for Coal Classification," Transactions, Am. Inst. Mining and Metallurgical Engrs., Coal Division, p. 125 (1932)."

From the above, it is evident that the bed moisture is the amount of water held within the particles of coal. It is not water on the surface that might come from overlying strata; it is not water held in chemical combination with either the coal substance or the mineral matter; it is water that can be expelled at 105°C, and it is water that is held within the pore spaces. The as-received basis on which coal analyses are frequently reported means that the analysis was made on coal in the condition in which it arrived at its destination. The moisture content of the coal may or may not be the same as it was before mining, and such may be true for the other determinations. Coal sampled from the "face" in the mine may be wet due to ground water and in that case the moisture content of the sample will be higher than the bed moisture. Thus, moisture in coal as-received is not necessarily the same as bed moisture but in the case of samples taken at a fresh face in the mine, the moisture content may be very nearly the same or identical with bed moisture. As stated above, the analyses used in this study of classification of coal were made on the as-received basis. The samples for these analyses were all taken from the face. Apparently there is not sufficient information available to judge the condition of these coals at the sampling face in regard to moisture. In any event the moisture content as given may be too high in many cases but in no case is it too low. It is believed that in general the moisture content of the coals used is a fair approximation of the bed moisture with the possible exception of the coals in the anthracite group.

Although the moisture content is much more important in the low-rank and noncoking coals and in that range is a significant factor in delineating coals, it also has some importance in the anthracite and meta-anthracite ranks. Variations in moisture content of coals of varying rank are shown in Fig 2 where it is seen that this constituent of coal is practically constant at 2 to 4 pct for semianthracite, low-volatile, medium-volatile, and most high-volatile A coals; but increases rapidly through the high-volatile B and C, subbituminous, and lignite coals, and less rapidly through the anthracite and meta-anthracite coals. That moisture is a fundamental property of the coal is indicated in Fig 3[3] where the relation between coal porosity and volatile matter is illustrated. Comparing Fig 2 with Fig 3, there is evidence of similarity of curves with the minimum in both cases occurring at about 20 pct dry, ash-free or mineral-matter-free volatile matter. Variation of heating value on the dry, mineral-matter-free basis with volatile matter is shown in Fig 4 and it is to be noted that it is also similar to Fig 2 and 3 although it bears an inverted relation to them. The maximum heating value occurs at about 20 pct dry, mineral-matter-free volatile matter. The source of data for Fig 2 and 4 is the same as for the classification curve in Fig 1.

MODIFICATION OF ASTM CLASSIFICATION

The study of the relation of coking properties to coal rank that will be described, resulted from an effort to use the standard method of classification for the prediction of the coking properties of coal. Many coals whose coking characteristics were known to the authors were superimposed on a chart similar to Fig 1. An examination of this plot indicated that in the high-volatile region of coals, for instance, there were cases where different coals having the same fixed carbon content on the dry, mineral-matter-free basis had quite different heating values on

the moist, mineral-matter-free basis as well as different coking characteristics. For example, some of the Illinois coals were found to have substantially the same fixed carbon as an Elkhorn seam and a No. 2 Gas seam coal, while two Pittsburgh seam coals from northern West Virginia had lower fixed carbon values than the Illinois coals, yet the Illinois coals were in the high-volatile B group and the others were of the high-volatile A rank. An examination of the coking behavior of these coals produced results that were equally anomalous. In other words, the standard classification method did not lend itself readily to the correlation of the coking properties of different coals.

The efforts of the authors, therefore, turned toward developing a method of classification that would be more adaptive to such correlation. A number of methods were studied and finally the system shown in Fig 5 was obtained. In this scheme of classification the ratio of the heating value of the coal in Btu per pound to percentage of volatile matter is plotted along the vertical logarithmic scale and percentage of volatile matter is plotted along the horizontal arithmetic scale. Both the heating value and volatile matter are on the mineral-matter-free basis and calculated with the use of the approximation formulas given above. The former is also on the moist basis while the volatile matter is on the dry basis. In Fig 5, nearly all the coals of Fig 1 were used so that the two methods are comparable.

Close examination of this plot reveals that the points representing the various coals fall very nearly on the line as shown from the anthracite group on the left and proceeding through the high-volatile A group on the right. The boundaries between the groups are distinct and according to the ASTM specifications. Below the high-volatile A group, the high-volatile B coals will be found to arrange themselves in a line just below the high-volatile A; the high-volatile C coals are found just below the high-volatile B coals, and so on down through the lignites.

While this plot achieved in a large measure the purpose of the study, the scattering of the coals below the high-volatile A group, even though fairly regular, was not entirely satisfactory. In seeking an explanation of the scattering, it appeared evident that the moisture content of the low-rank coals was accountable. By adding the moisture content to the dry, mineral-

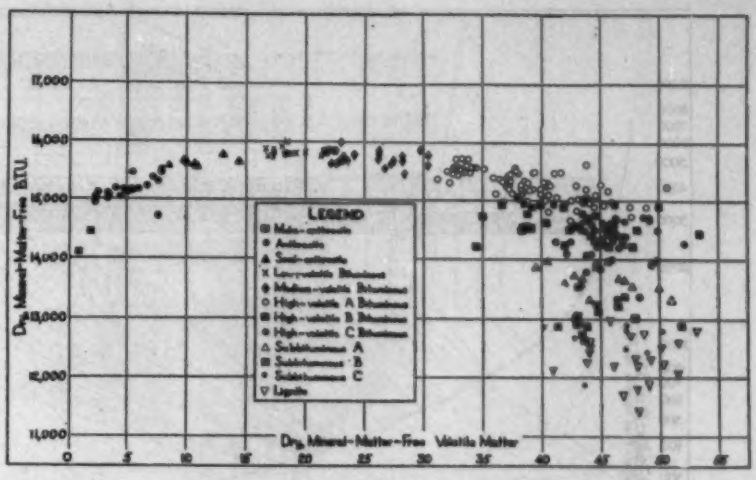

FIG 4—Variation of calorific value of coal with rank. (Data from U.S. Bureau of Mines *R.I.* 3296 R.)

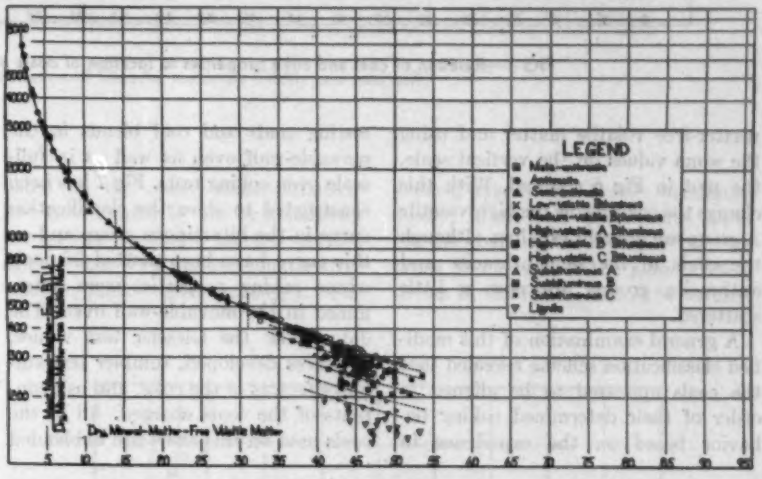

FIG 5—Classification of typical coals of the United States. (Data from U.S. Bureau of Mines *R.I.* 3296 R.)

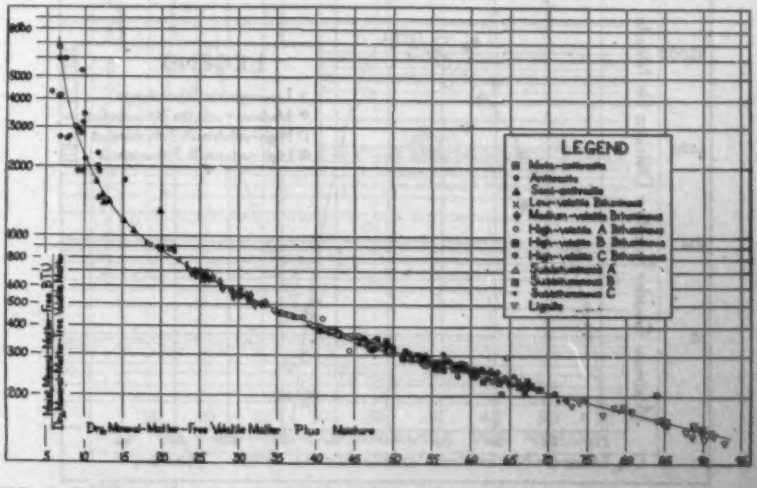

FIG 6—Classification of typical coals of the United States. (Data from U.S. Bureau of Mines *R.I.* 3296.)

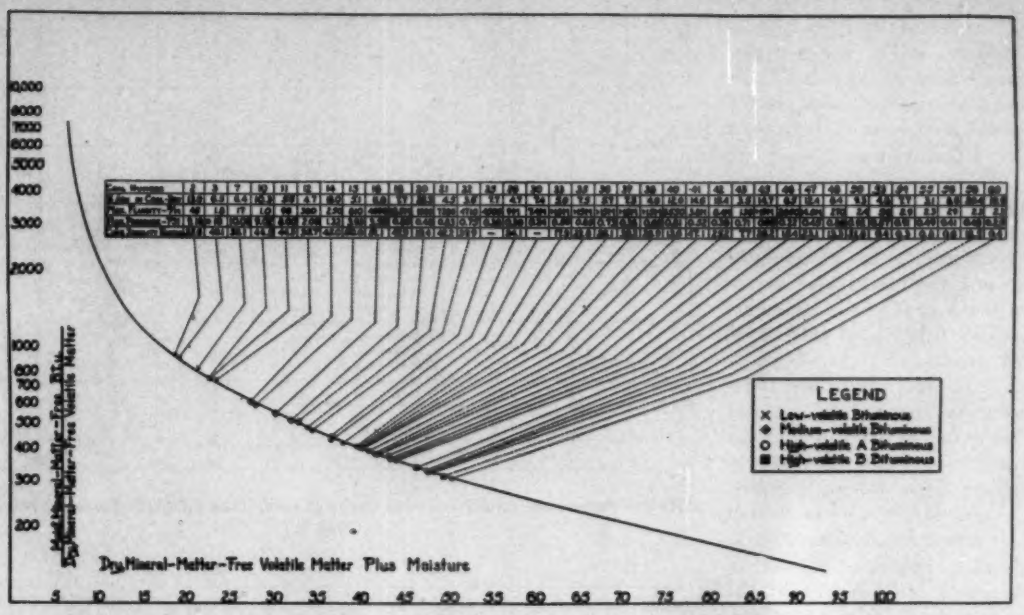

FIG 7—Relation of coal and coke properties to location of coals on classification curve.

matter-free volatile matter and using the same values on the vertical scale, the plot in Fig 6 resulted. With this change the coals below the high-volatile A group are brought into line, although the coals in the semianthracite and anthracite groups are now a little scattered.

A general examination of this modified classification scheme revealed that the coals appeared to be aligned in order of their determined coking behavior based on the experience in testing coals and coal blends in the movable-wall oven as well as in full-scale oven coking tests. Fig 7 has been constructed to show the classification curve in the bituminous group and on this curve have been plotted 36 coals whose coking properties were determined in the movable-wall oven. The data show the Gieseler test values, pressures developed, tumbler test stability factors of the coke, and ash contents of the coals charged. All of the coals used on this chart are unblended coals. Although it is evident that there are a number of cases where the sequence of values of the coking behavior is not continuous in traversing the curve of coal rank, there is a very definite trend. A number of the cases where coals appear to be out of sequence can possibly be explained by their ash content. In other cases, such exceptions might be explained by variation in petrographic constituents and undoubtedly the inaccuracies of the testing methods available can account for other variations. It should be said, however, that the testing of the coals and the cokes produced was done by the standard methods. Every effort was made to conduct the tests as accurately as possible. However, it must be acknowledged that present procedures for measuring the physical properties of coke require further refinement before satisfactory accuracy can be obtained.

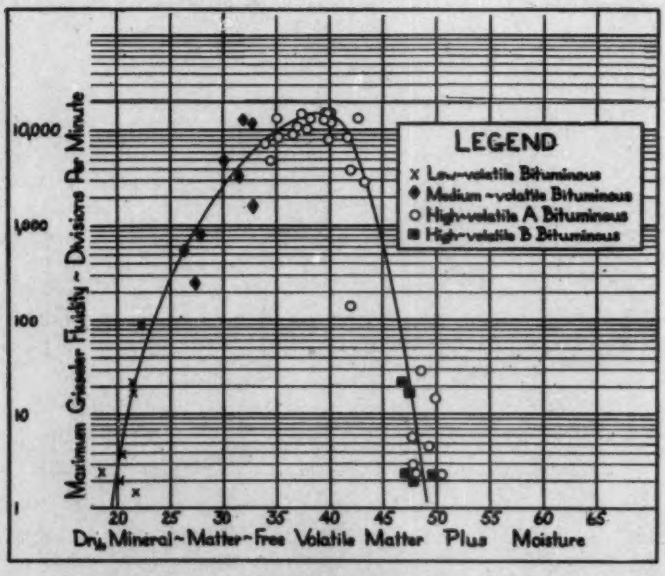

FIG 8—Relation of maximum Gieseler fluidity to coal rank.

Correlation of Coking Behavior with Coal Rank

GIESELER MAXIMUM FLUIDITY

A more detailed study was therefore undertaken to establish any relation between the classification scheme developed and the coking behavior of

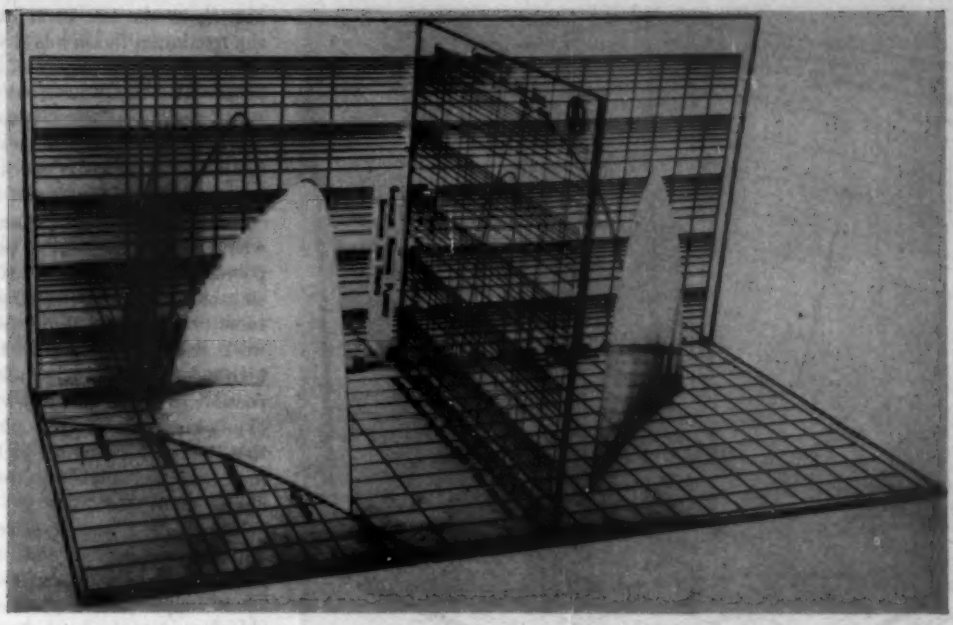

FIG 9—Correlation of Gieseler maximum fluidity and coal rank.

coals. The Gieseler test[10] for the plastic properties of coals has been used by Koppers Company, Inc., Research Department for about ten years and the data thus accumulated were first applied. Fig 8 presents the relation of the maximum Gieseler fluidity of coals to coal rank as indicated by the dry, mineral-matter-free volatile matter plus moisture. Only the coking coals are shown since the others do not exhibit fluid properties when heated. A relatively good correlation is evident with the points falling on or near a hump-shaped curve. Low-volatile coals exhibiting low fluidities are found at the lower left end of the curve. Further up the curve to the right are found medium-volatile coals and at or near the peak of the curve are found the highly-fluid high-volatile A coals. Along the downward portion of the curve to the right are found less fluid high-volatile A and B coals or coals containing higher amounts of oxygen. The peak of the curve is found to correspond to approximately 40 pct dry, mineral-matter-free volatile matter plus moisture. In other correlations to be shown later, it will be seen that the peaks also occur at about this same value.

Fig 9 is a three-dimensional plot of Gieseler data in two quadrants. In the quadrant on the left, the plane of the base contains the classification curve based on the moist, mineral-matter-free Btu divided by the dry, mineral-matter-free volatile matter plotted against the dry, mineral-matter-free volatile matter plus moisture. The vertical axis is the Gieseler maximum fluidity in divisions per minute. Data for the coals so far tested of various ranks lie on or near the plane surface shown in the figure. It will be evident

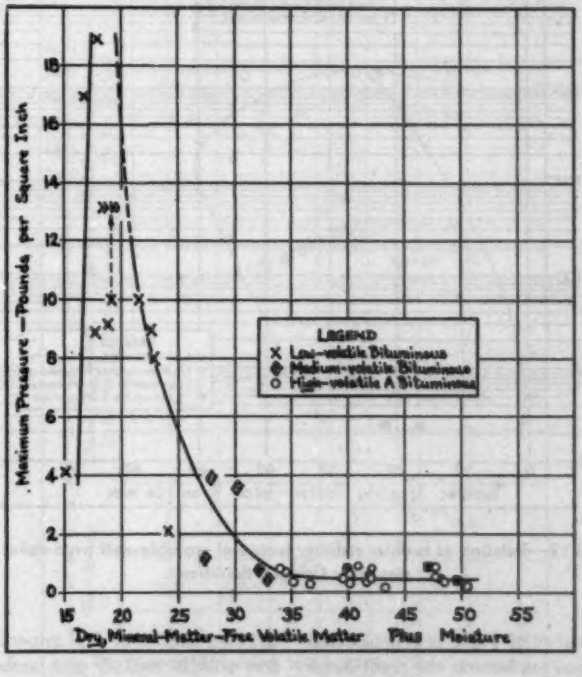

FIG 10—Relation of movable-wall oven maximum pressures to coal rank.

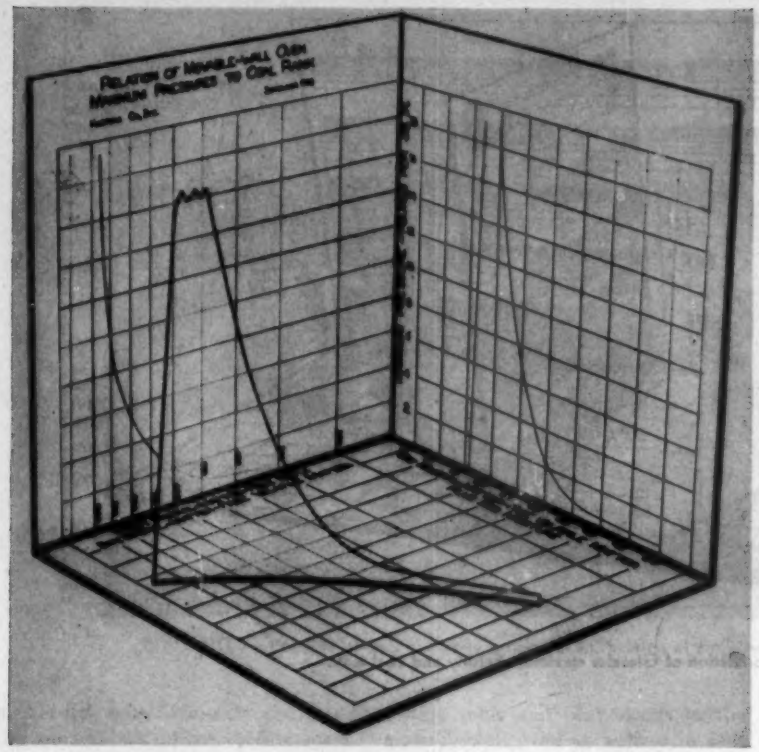

FIG 11—Relation of movable-wall oven maximum pressures to coal rank.

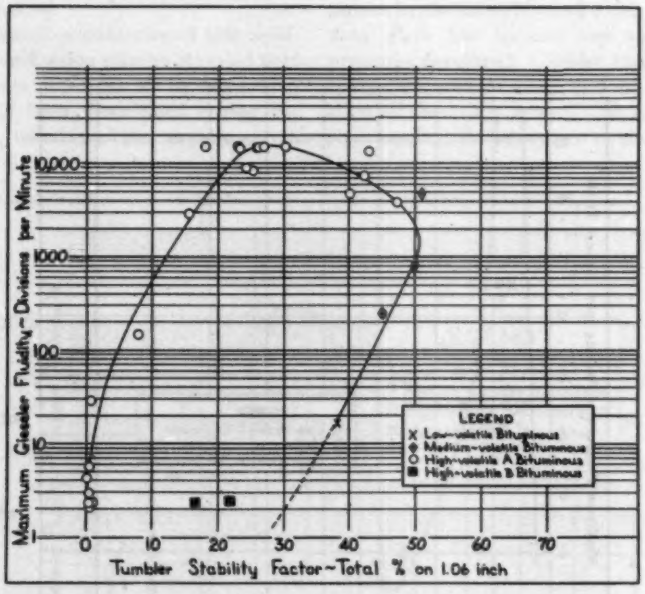

FIG 12—Relation of tumbler stability factors of movable-wall oven cokes to maximum Gieseler fluidities.

that the curve of Fig 8 is a projection of this plane surface on the right hand vertical coordinate plane. The quadrant to the right contains a base having the coordinates of dry, mineral-matter-free volatile matter plus moisture and temperature in degrees Centigrade of the maximum fluidity. The vertical axis as in the left hand quadrant is the maximum fluidity in divisions per minute. A similar plane surface erected in this right hand quadrant shows the relation of these three last-named variables. This relation will be useful in a further study of the work of Marquard[11] in which he stated that coals of different softening temperatures when blended before coking produce poorer coke than when coals of substantially the same softening temperatures are blended. While Marquard's work referred specifically to various high-volatile coals, it should now be possible to study this idea with blends of any kinds of coal. It should be noted that the coals having the highest maximum fluidity attain this condition at approximately 435°C.

The value of the Gieseler plastometer lies in the fact that significant information can be obtained from a small sample (4 to 5 g) of coal which together with the analysis heretofore had yielded an estimate of the coking behavior of the coal. In the past, the appraisal of unknown coals has been made almost entirely by relatively large-scale tests. Box tests, for example, requiring about 60 lb of coal were considered to be about the smallest test procedure that would give an indication of the type of coke to be expected. After box tests, a full-scale oven test was frequently required. The Gieseler test has not eliminated this procedure but it has been the source of information from which a more intelligent and practical approach to the selection of coals could be made. The fact that the maximum fluidity can be correlated with values obtained from proximate analyses may lead ultimately to the use of proximate analyses as a guide to the selection of coals. However, there is still much to be learned from the Gieseler test about the mechanism of coke formation.

PRESSURES DEVELOPED BY COAL

The next correlation attempted was with the maximum pressures developed by coals in the movable-wall oven. Fig 10 contains data on unblended coals, and while the information at hand is far from complete, there is shown a very definite trend in this factor with the rank of coal. All of the high-volatile A coals shown exerted a pressure of less than 1 psi. Several of the medium-volatile coals exerted more than 1 psi and are shown to go up as high as 4 lb as the rank increased

toward the low-volatile group. Two of these coals contained a high percentage of ash and the latter acting as an inert had reduced the pressure-developing properties of the coals resulting in the points falling below the curve. The low-volatile coals exerted higher and higher pressures as the rank increased and finally reached a maximum whose value is still undetermined. This is due, in part, to the inability to measure these high pressures under the same conditions that are used for measuring the pressures of the lower rank coals. As the rank increased still further the pressure dropped off. Several English coals are shown in this area and one with a dry, mineral-matter-free volatile matter plus moisture of about 15 pct was found to exert a pressure of just 4 psi. This behavior with changing rank appears to have some reason in fact, for as the rank increases to the semi-anthracite group and the fluidity decreases, such coals can conceivably produce little or no pressures. Much more work needs to be done in this area as well as in medium-volatile group near the low-volatile boundary. No satisfactory relation has yet been found that will indicate the pressures exerted by blends of coal.

Fig 11 is a three-dimensional plot of pressures developed in the movable-wall ovens. The plane of the base contains the classification curve and plotted vertically is the pressure developed in pounds per square inch. It is evident that the preceding figure (Fig 10) is really a projection of the curved surface rising out of the base projected on the right hand coordinate plane.

MAXIMUM FLUIDITY AND STABILITY FACTOR

Fig 12 presents the relation between the Gieseler maximum fluidity of unblended coals and the tumbler test stability factors of the resultant cokes. Although the data are not complete, there is sufficient information on hand to show that the curve fixes the approximate relationship. Low-volatile coals having a low maximum fluidity produce coke of higher stability factor than the high-volatile coals having substantially the same maximum fluidity. The medium-volatile coals and the high-volatile A coals near the medium-volatile group produce coke of the highest stability factor. Coals of the highest maximum fluidity produce cokes of intermediate stability factor.

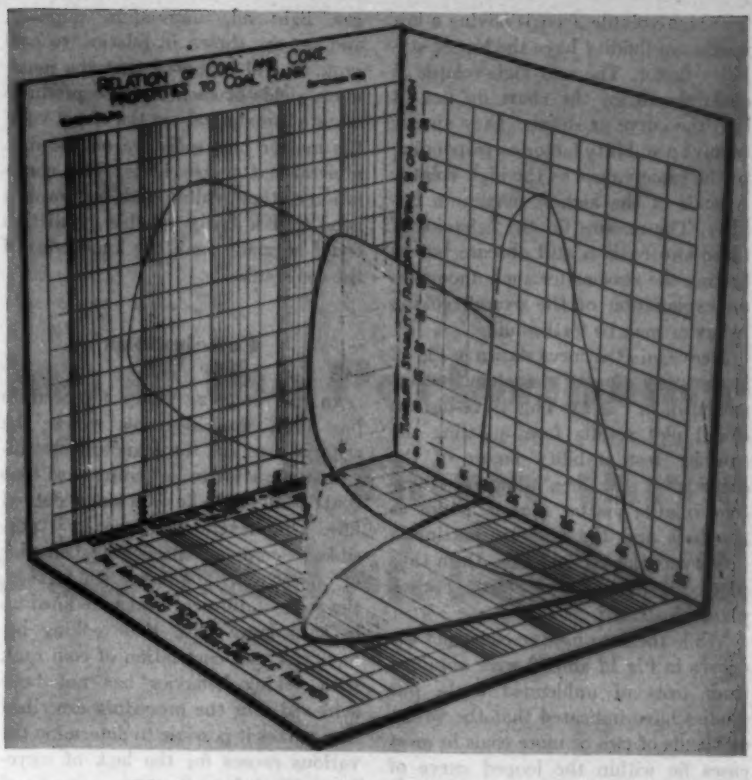

FIG 13—Relation of coal and coke properties to coal rank.

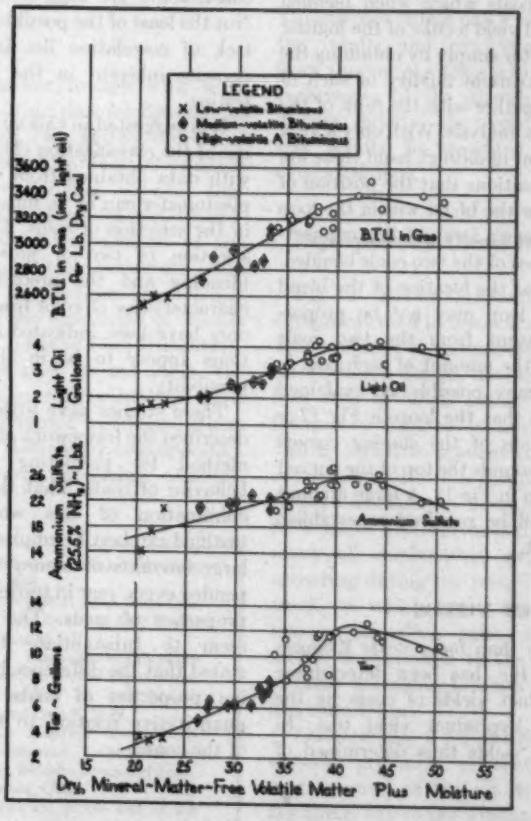

FIG 14—Relation of byproduct yields to coal rank.

The high-volatile A coals having a low maximum fluidity have the lowest stability factor. The two high-volatile B coals shown on the chart do not fit into the curve as shown. They appear to have a stability factor corresponding to the same values as the high-volatile A coals of the highest maximum fluidity. The reasons for the behavior of these two coals is still obscure but in finding the reasons for such anomalies an explanation of the overall coking behavior may be forthcoming.

Here again the curve shown in Fig 12 is the projection of the plane surface as shown in Fig 13. This three-dimensional plot results from plotting the tumbler test stability factor of the cokes with the dry, mineral-matter-free volatile matter plus moisture as the base and the Gieseler maximum fluidity as the vertical axis. Data thus plotted lie along the top of the curved plane illustrated.

While the data used to construct the curves in Fig 12 and 13 were obtained from tests of unblended coals, our studies have indicated that the values of blends of two or more coals in most cases lie within the looped curve of Fig 12. If further data confirm this hypothesis it will then be possible to select two coals which when blended together will yield a coke of the highest stability factor simply by obtaining the Gieseler maximum fluidity of each of the coals together with the rank of the coal from its analysis. With only a very small amount of data at hand there are further indications that the position of the point for the blend within the loop lies on or near a straight line connecting the values of the two coals blended. The fact that the location of the blend within the loop may not be proportionally distant from the two coals blended to the amount of each coal in the blend may possibly be explained by the fact that the loop in Fig 12 is the projection of the sloping curved plane that bounds the top of the curved plane shown in Fig 13. A large amount of work will be required to establish this concept.

BYPRODUCT YIELDS

For more than forty years Koppers Company, Inc. has been determining the byproduct yields of coals by the laboratory byproduct yield test. In Fig 14 the yields thus determined of gas, light oil, ammonium sulphate, and tar are shown in relation to coal rank. It will be noted that the maximum yields of each of these products occur approximately at the 40 pct volatile matter plus moisture on the dry, mineral-matter-free basis. The correlation of these values with coal rank is good and a fairly reliable prediction could be made from the knowledge of the coal rank.

Conclusions

In the foregoing study a procedure has been described by which the criteria used for the standard classification of coals can be arranged so that coals of all ranks lie on a single curved line. This development makes it possible to compare the coking behavior of one coal with another. In general, the coals of different ranks are lined up in accordance with their coking behavior. Exact correlation of coal rank with coking behavior has not been achieved, but the procedure described now makes it possible to determine the various causes for the lack of correlation. Such factors as ash content and differences in amount of petrographic constituents are suggested as causes. Not the least of the possible reasons for lack of correlation lies in the inaccuracies inherent in the methods of testing.

It is suggested in this study that the use of the classification charts together with data obtained from the Gieseler plastometer can be an important guide in the selection of coals. Although the selection of two or more coals for blending and the prediction of the characteristics of coke from the blend only have been indicated such predictions appear to be in the realm of possibility.

These studies have little more than described the framework of a simplified method for predicting the coking behavior of coals. That the successful culmination of this work is well justified can best be emphasized by the large amounts of money that are expended every year in testing the coking properties of coals. The correlations seem to substantiate the premise stated that the difference between coking properties of coals bears some quantitative relation to the analyses of the coals.

Acknowledgment

The authors are indebted to Mr. Fred Denig, Vice President and Director of Research, Koppers Company, Inc., for his continued interest and helpful suggestions and for permission to publish the information presented. We also wish to thank Messrs. J. F. Farnsworth, J. A. Hahn, J. K. McHugh, and H. B. Smith, Jr. for their assistance in obtaining data and preparation of the figures.

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An Evaluation of the Performance of Thirty-three Residential Stoker Coals

By JAMES B. PURDY* and HARLAN W. NELSON*

The great majority of stokers used in residential heating installations are of the clinkering type. Because of inherent characteristics of the underfeed combustion process as it occurs in these small stokers, and because of the wide range of operating conditions encountered in residential heating schedules, not all bituminous coals can be used successfully in residential underfeed units. The problem of providing consumer satisfaction has been attacked by the joint efforts of the stoker manufacturers and the coal industry; the stoker manufacturers have been concerned with improving their product to permit the use of a wider range of coals, and coal operators have made every effort to market selected and better prepared coals. In view of the wide range of physical characteristics and chemical properties of bituminous coals, the problems are real and are receiving increased attention.

Some twelve years ago a laboratory method was developed at Battelle Memorial Institute for the relative evaluation of coals for use in residential underfeed stokers. The method was developed in conjunction with work being done at that time for Bituminous Coal Research, Inc., and for individual coal producers, to obtain information on the characteristics of coal that determined the successful performance of the small stoker. Details of the equipment and test procedure were given by R. A. Sherman[1] in a paper presented at the University of Illinois Fifth Short Course in Coal Utilization.

During the past several years, the evaluation of over forty different coals has been determined by investigations sponsored at Battelle by coal operators in the eastern producing districts.

White Sulphur Springs Meeting, November 1948.

Paper presented at Joint Fuels Conference of Coal Division AIME and Fuels Division ASME.

TP 2568 F. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before July 30, 1949. Manuscript received Dec. 7, 1948.

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[1] References are at the end of the paper.

The purpose of this paper is to compare the results and conclusions derived from the various tests with marketing experiences of the operators, and to discuss briefly the relationship of test results to standard laboratory determinations.

Equipment Used in Tests

Fig 1 shows schematically the equipment used in the evaluation tests. The stoker is a commercial model of conventional design, having a nominal feed rate of 30 lb per hr. A special laboratory furnace is used, which is tightly sealed to prevent the entrance of secondary air. The complete stoker-furnace assembly is mounted on platform scales which permits the measurement of combustible weight losses occurring during the test. The furnace stack, in which are a thermocouple, flue-gas sampling tube, and smoke meter, is connected to the laboratory exhaust breeching through a frictionless connection.

Through use of an auxiliary blower system, provision is made for the measurement of the air delivered by the stoker fan to the windbox. A water seal in the duct work eliminates inter-

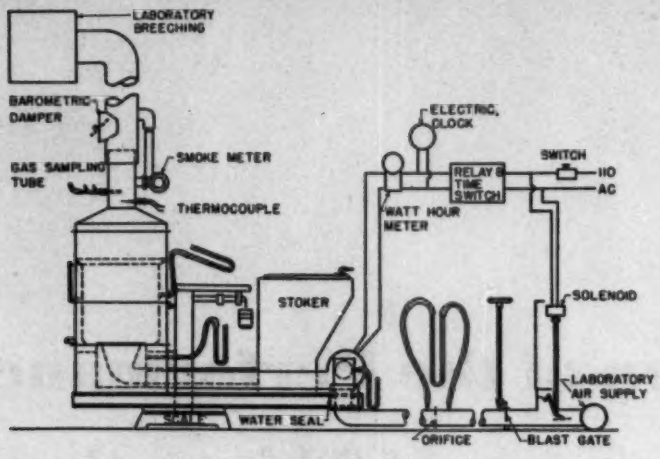

FIG 1—Diagram of test equipment.

ference with the process of weighing the stoker-furnace unit. A barometric damper is located in the furnace stack to assist in maintaining a constant draft. The damper is adjusted to give a draft of 0.05 in. of water during "off" periods of the stoker.

Smoke concentrations are measured by means of a smoke-density meter. The meter consists of a light source and photoelectric cell between which a sample of the flue gases is drawn at a constant rate. Variations in the absorption of light by smoke varies the output of the photoelectric cell, and the voltage is recorded continuously on a recording potentiometer.

Appropriate instruments are provided for control of hold-fire operation, for recording stoker running time and power consumption, and for measurement of overfire and windbox pressures. Flue-gas analyses are made at appropriate intervals throughout the test.

Test Procedure

At the start of the test, the hopper is filled with the coal to be tested, and the stoker is run until the retort is filled and coal is level with the top of the tuyeres. The feed of coal is then stopped, and 1 lb of charcoal is placed on top of the coal. Just prior to lighting the fire, one-half pint of kerosene is poured over the charcoal. The charcoal is then ignited and for a period of 5 min no coal is fed, but the stoker fan is permitted to run. At the end of 5 min the coal feed is turned on and allowed to run for the balance of the hour.

The air control on the stoker is set to furnish air at a rate 30 pct in excess of that theoretically required by the rate at which the coal under test is being fed. Any correction necessary to obtain this rate of air delivery is made during the first hour by means of the stoker air control. No further adjustment of the air is made throughout the test.

The schedule of operation for an evaluation test is designed to simulate the operation of a stoker under conditions of varying heat demand, as would be expected during a typical heating season. The test is normally of five days' duration. During the first and second days, operation consists of alternate periods of continuous and hold-fire operation; alternate periods of intermittent and hold-fire operation occupy the third day, and the fourth day consists wholly of hold-fire operation. Intermittent operation is

Table 1 . . . Analyses of Coals Tested

Test Number	Bed	State	Nominal Size, In.	Proximate Analysis, Per Cent					Sulphur, Per Cent	Calorific Value, Btu per Lb	Ash-softening Temp., °F	Free-swelling Index, Number
				Moisture	Volatile Matter	Fixed Carbon	Ash	Total				
1	Wallins	Ky.	1¼ × 0	2.5	40.1	51.8	5.6	100.0	0.8	13,630	2,560	
2	Wallins	Ky.	1¼ × ¾	2.8	40.0	52.4	4.8	100.0	0.8	13,700	2,250	
3	Upper Freeport	Pa.	¾ × ¼	1.1	33.8	56.0	9.1	100.0	1.6	13,809	2,360	
4	Pittsburgh	W. Va.	1 × 10 mesh	3.0	37.5	53.3	6.2	100.0		13,380	2,380	4.5
5	No. 2 gas	W. Va.	1 × 10 mesh	0.9	34.6	61.2	3.3	100.0	0.8	14,760	2,290	
6	Taggart	Va.	¾ × 0				3.1				2,210	8.5
7	Taggart	Va.	¾ × 0				4.5				2,310	8.5
8	Chilton	W. Va.	¾ × ⅜	1.1	31.9	58.5	8.5	100.0			2,700	7.0
9	Sewell	W. Va.	¾ × ¼	0.2	23.2	72.6	4.0	100.0		15,140	2,440	
10	Pittsburgh	W. Va.	¾ × ⅜	2.7	38.1	54.4	4.8	100.0		14,050		8.0
11	Elkhorn No. 3	Ky.	1 × ¼	1.6	37.2	54.6	6.6	100.0		13,920		6.0
12	Elkhorn	Ky.	1 × ⅜	0.9	35.4	60.5	3.2	100.0			2,630	6.0
13	Elkhorn	Ky.	⅜ × ¼	1.6	32.9	59.8	5.7	100.0			2,660	5.5
14	Elkhorn	Ky.	1 × ¼	1.4	34.8	60.0	3.8	100.0			2,530	6.0
15	Elkhorn	Ky.	1 × 50 mesh	1.3	34.1	59.5	5.1	100.0			2,670	6.0
16	Mason	Ky.	1 × ¼	0.6	37.8	56.4	5.2	100.0	1.6	14,160	2,420	5.0
17	Mason	Ky.	1 × ¼	1.7	36.3	58.3	3.7	100.0		14,089	2,530	4.0
18	No. 11	Ky.	¾ × 48 mesh				6.1				2,200	4.0
19	High Splint	Ky.	1 × ⅝	0.7	39.0	52.5	7.8	100.0	1.0	13,400	2,730	3.5
20	Blue Gem	Ky.	1¼ × ⅜	1.9			6.0		1.6	13,680		3.5
21	No. 2 gas	W. Va.	1 × ¾	1.7	35.7	55.7	6.9	100.0	1.4	13,951	2,680	8.5
22	(Unknown)	Ind.	¾ × ⅛	7.5	41.0	44.0	7.5	100.0	3.0	12,397	2,190	
23	Hazard No. 4	Ky.	1 × ¾	1.9	37.6	55.2	5.3	100.0	0.8	13,961	+2,700	5.0
24	Hazard No. 4	Ky.	1 × ¾	2.2	37.5	55.2	5.1	100.0	0.9	13,940	2,410	4.5
25	Elkhorn No. 4	Ky.	1 × ¾	1.1	38.5	55.6	4.8	100.0	1.0	13,970	2,500	5.0
26	Hernshaw	W. Va.	1 × ¾	0.8	32.3	63.6	3.3	100.0	0.7	14,690	2,750	7.5
27	Rich Mountain	Tenn.	1¼ × ¾	1.3	35.7	56.1	6.9	100.0	1.0	13,730	+2,800	4.5
28	Shannon	Va.	1¼ × ¾	1.9	33.6	53.4	11.1	100.0	1.1	13,283	2,280	6.5
29	No. 2	Ky.	1 × ¾	1.3	37.3	46.5	14.9	100.0	5.1	12,470	2,310	3.5
30	No. 7	Ky.	1 × ¼	6.0	38.9	51.4	4.6	100.0	1.8	12,780	2,190	4.0
31	Powellton	W. Va.	1 × ¾	1.8	35.5	57.1	5.6	100.0	1.1	14,330	2,420	8.5
32	Hazard No. 4	Ky.	1 × ¼	1.2	38.0	55.2	4.7	100.0	2.6	14,130	2,340	5.0
33	Clintwood	Ky.	1 × 4 mesh	0.4	29.1	63.9	6.6	100.0	1.1	14,360	2,700	8.0

scheduled for the whole of the fifth day.

For the parts of the schedule indicating continuous operation, the stoker is started and stopped manually. For hold-fire operation, the stoker is started once each hour by a timer control and is stopped by a potentiometer controller when the temperature of the flue gases has reached 20 mv, or approximately 900°F. During intermittent operation, the stoker is manually started at intervals of 20 min, the duration of each period of operation being determined by the time required to burn one-third of the quantity of coal necessary to release 187,500 Btu per hour. The stoker is stopped when the required weight loss is attained, and remains idle until the start of the next 20-min period.

As a test is always started with a completely clean hearth, the fuel bed is not disturbed until the evening of the fourth day, when a probe is made with clinker tongs to remove any clinker from the fuel bed. At the conclusion of the test the fire is quenched with water and the fuel bed is removed from the furnace. Pieces of clinker larger than 3 in. in size, and all coke retained on a 1-in. round-hole screen, are separated from the fuel bed and weighed.

Bases for Evaluation

In evaluating a stoker coal certain tangible factors must be selected on which a judgment of the coal may be based. These factors are chosen because of their importance in determining the degree of satisfaction that the consumer will obtain in using the coal, from the standpoint of manual attention, uniformity of heating, comfort, cleanliness, and cost. The main factors used as a basis for evaluation in this test are:

1. Initial ignition.
2. Clinker formation.
3. Coke formation.
4. Rate of fuel consumption during hold-fire operation.
5. Resistance of fuel bed to the flow of air.
6. Uniformity of burning during intermittent operation.
7. Smoke concentration.

Coals Tested

Over 40 tests have been conducted, and their results evaluated by the method described. A few of these were short-term tests consisting of only the first and last days of operation. Of the 5-day tests, the results of 33 are included in the present study. With one exception, the coals were all high volatile, and were largely from the eastern producing districts.

Table 1 presents the laboratory-determined characteristics of the coals used in these tests. In addition to the proximate analysis, the sulphur, calorific value, ash-softening temperature, and free-swelling index are given. The nominal size of the coal is indicated.

Results of Tests

At the conclusion of each test, average values are determined for components of the flue-gas analysis, pressure measurements, smoke concentrations during "off" periods, and length of hold-fire running periods. These and other directly determined values are then subjected to analysis.

Table 2 lists values of many of these data for the coals tested. When reporting the results of tests, in addition to listing data shown in the table, plots of some groups of data are made for purposes of showing their degree of variability during the course of the test. Wide variations in the length of operating times during intermittent operation, for example, are indicative of excessive coke-tree formation, dirty fuel bed, or general lack of response to heat-demand operation, but average values alone do not indicate this variability. Supplementary data and information, including photographs and visual observation of the fuel bed, strength of the clinker formed, and the height and type of coke trees, are not subject to numerical evaluation but are useful in making an overall interpretation of results.

The ratings of initial ignition shown in Table 2 represent approximate evaluations of the curve of flue-gas temperatures recorded during the first hour of the test. The word descriptions shown furnish an initial indication of the ignitibility and coking characteristics of the coal under test. The curves for initial ignition are discussed more fully in a later section of the paper.

For the coals tested, the range of values for the drop in pressure through the fuel bed was from 0.52 to 1.54 in. of water. The values are indicative of the general condition of the fuel bed, and any coal that has a fuel bed resistance greater than 1.5 in. of water is considered to give difficulty in burning. Of the coals tested, only one showed values above this maximum.

As previously stated, during hold-fire operation, the stoker is started once each hour and allowed to run until a stack temperature of 900°F is reached. Under this procedure, the

Table 2 . . . Summary of Results from Evaluation Tests

Test Number	Bed	State	Initial Ignition	Pressure Drop through Fuel Bed, In. H_2O	Hold-fire Time, Min.	Smoke Concentration, Average	Coke in Fuel Bed, End of Test, Lb	Clinker Removed, Per Cent of Ash Released
1	Wallins	Ky.		1.16		11.8	4.1	16.8
2	Wallins	Ky.		0.86		13.6	2.6	27.2
3	Upper Freeport	Pa.		1.54	4.5	27.6	14.5	24.0
4	Pittsburgh	W. Va.		1.08	5.4	21.6	9.6	47.6
5	No. 2 gas	W. Va.		0.84	4.1	15.6	16.5	27.9
6	Taggart	Va.	Satisfactory	0.76	4.3	12.0	4.3	13.4
7	Taggart	Va.	Very satisfactory	0.91	5.3	16.7	9.6	21.0
8	Chilton	W. Va.	Satisfactory	1.26	6.8	14.0	3.8	14.0
9	Sewell	W. Va.	Poor	1.19	4.2	7.5	16.1	25.3
10	Pittsburgh	W. Va.	Poor	0.79	5.4	33.4	4.9	16.3
11	Elkhorn No. 3	Ky.	Very satisfactory	1.01	5.8	14.7	10.8	28.4
12	Elkhorn	Ky.	Very satisfactory	0.52	4.6	14.7	8.4	12.5
13	Elkhorn	Ky.	Satisfactory	0.85	4.3	9.4	16.4	39.8
14	Elkhorn	Ky.	Satisfactory	0.65	4.2	10.9	7.8	26.2
15	Elkhorn	Ky.	Satisfactory	0.86	4.3	14.4	19.5	36.6
16	Mason	Ky.	Poor	1.05	5.3	15.8	9.1	37.1
17	Mason	Ky.	Very satisfactory	0.72	4.1	10.6	4.1	21.3
18	No. 11	Ky.	Satisfactory	0.97	4.3	17.9	8.4	31.0
19	High Splint	Ky.	Very satisfactory	0.77	4.8	5.0	3.6	17.8
20	Blue Gem	Ky.	Very satisfactory	0.77	4.9	12.8	13.7	53.3
21	No. 2 gas	W. Va.	Poor	0.81	6.0	19.0	19.0	16.2
22	(Unknown)	Ind.	Satisfactory	0.68	6.3	16.0	6.6	21.2
23	Hazard No. 4	Ky.	Satisfactory	1.09	5.4	12.0	7.2	23.5
24	Hazard No. 4	Ky.	Very satisfactory	0.97	4.5	16.0	7.4	14.9
25	Elkhorn No. 4	Ky.	Satisfactory	0.82	5.7	20.0	8.6	14.0
26	Hernshaw	W. Va.	Very satisfactory	0.78	4.6	11.0	5.4	11.6
27	Rich Mountain	Tenn.	Poor	1.26	6.3	21.0	11.9	17.5
28	Shannon	Va.	Poor	0.90	5.9	15.0	7.5	26.0
29	No. 2	Ky.	Poor	1.28	7.1	17.0	42.8	45.4
30	No. 7	Ky.	Very satisfactory	0.63	5.1	15.0	3.9	18.4
31	Powellton	W. Va.	Poor	0.93	5.7	22.0	17.5	36.0
32	Hazard No. 4	Ky.	Satisfactory	0.63	5.2	20.7	10.2	28.6
33	Clintwood	Ky.	Poor	1.04	6.2	18.6	15.5	30.5

time of operation is governed by characteristics of the coal. A cut-off temperature of 900°F may be too high, as many coals will be capable of holding the fire at a lower temperature; however, time would not permit trial of minimum settings, and the results may be compared on a relative basis. For purposes of evaluation, coals having time values below 5 min per hr are considered satisfactory. The range of values shown in Table 2 was from 4.1 to 7.1 min per hr. Values of less than 5.0 min were determined for 14 of the coals tested, and 6 of the coals required operating periods averaging over 6.0 min.

The values of smoke concentration were determined as the average density of smoke during the "off" periods of intermittent operation. Only coals showing average values above 20 on the logarithmic scale of densities would be apt to produce troublesome smoke during "off" periods, as combustion of volatile gases and dilution of smoke by air admitted over the fire would substantially reduce the smoke concentrations in a residential heating installation.

Values of the weight of coke remaining in the fuel bed at the end of a test give a numerical indication of the extent of coke formation and the depth of the fuel bed. Comparisons are significant only when all tests are run with closely similar percentages of excess air. The range of values found during the present tests was from 2.6 to 42.8 lb; 8 of the coals produced 15.0 lb of coke or more.

The measure of clinker formation represented by data listed in Table 2 is the percentage of the total ash released during the test, which is formed into clinker. Percentage values of 20 pct or more are considered to indicate satisfactory clinker formation.

Table 3 . . . Summary Information Regarding Conclusions from Evaluation Tests and Marketing Experience of Coal Operators

Test Number	Bed	State	Classification of Laboratory Test[a]	Reason for Classification	Experience in Marketing by Sponsor Operator
1	Wallins	Ky.	S		No information
2	Wallins	Ky.	S		No information
3	Upper Freeport	Pa.	U	Strong coke-tree formation	Does not market residential stoker coal
4	Pittsburgh	W. Va.	S		No information
5	No. 2 gas	W. Va.	S		Marketed successfully
6	Taggart	Va.	S		Marketed successfully
7	Taggart	Va.	S		Marketed successfully
8	Chilton	W. Va.	U	Poor clinker formation and hold-fire characteristics	Not marketed for use in residential stokers
9	Sewell	W. Va.	S		No information
10	Pittsburgh	W. Va.	S−	Sluggish ignition, length of hold-fire periods above average, low clinker formation	No information
11	Elkhorn No. 3	Ky.	S		Marketed successfully
12	Elkhorn	Ky.	S−	Poor clinker formation	No information
13	Elkhorn	Ky.	S		No information
14	Elkhorn	Ky.	S		No information
15	Elkhorn	Ky.	S		No information
16	Mason	Ky.	S		Marketed successfully, principally in northern markets
17	Mason	Ky.	S−	Relatively poor clinker formation	Facilities not yet available for preparation of stoker coal
18	No. 11	Ky.	S		Marketed successfully
19	High Splint	Ky.	S−	Clinker formation questionable	Marketed successfully
20	Blue Gem	Ky.	S		Mine now closed, plan to prepare stoker coal when re-opened
21	No. 2 gas	W. Va.	S−	Sluggish ignition, strong coke formations, clinker formation low	No information
22	(Unknown)	Ind.	S		New mine, upon completion of tipple will market stoker coal
23	Hazard No. 4	Ky.	S		Small mine, output not yet justifies preparation of stoker sizes
24	Hazard No. 4	Ky.	S−	Inadequate clinker formation	Has been marketed successfully
25	Elkhorn No. 4	Ky.	S−	Inadequate clinker formation	No stoker coal prepared for market
26	Hernshaw	Ky.	S−	Poor clinker formation	Preparation facilities lacking, no stoker sizes prepared
27	Rich Mountain	Tenn.	U	Sluggish ignition and hold-fire characteristics, high in coke formation, low in clinker formation	Only run-of-mine being marketed
28	Shannon	Va.	U	Sluggish ignition and hold-fire characteristics, high ash content	No stoker sizes prepared
29	No. 2	Ky.	U	Poor ignition and hold-fire characteristics, sluggish fuel bed, high ash content, heavy coke formation	Decision made not to prepare stoker coal
30	No. 7	Ky.	S		Stoker sizes not being prepared at present time
31	Powellton	W. Va.	S		Stoker sizes not being prepared at present time
32	Hazard No. 4	Ky.	S		New tipple under construction, stoker sizes will be prepared
33	Clintwood	Ky.	S		Equipment for preparation of stoker sizes not yet installed

[a] S, Satisfactory. S−, Not completely satisfactory. U, Unsatisfactory.

Relation of Conclusions Drawn from Test Results to the Marketing Experience of Operators

From consideration of the test results, conclusions are drawn with regard to the degree of satisfaction to be expected if the coal under test were to be marketed. Many of the coals tested could immediately be classed as satisfactory or unsatisfactory beyond reasonable doubt; others were deficient in performance by one or more criteria, and were given an intermediate or limited classification.

Table 3 lists the coals tested, their classification based on test results, the principal reasons for the classification, if unsatisfactory or not completely satisfactory, and the marketing experience of the coal operators sponsoring the tests. Of the 33 coals tested, 20 were determined to be satisfactory for use in residential stokers, 5 were listed as being unsatisfactory, and 8 were given limited ratings.

The chief cause for limited ratings was inadequate clinker formation. Although satisfactory in other respects, use of coals in the restricted classification might be expected to result in complaints from consumers because of the accumulation of unfused ash in the furnace. Such coals might be marketed successfully in the northern regions, but might prove unsatisfactory in regions where climatic conditions require frequent and prolonged operation at low or hold-fire rates. Other factors adversely affecting the evaluation of the coals were excessive coke-tree formation, sluggish ignition and response during intermittent operation, high ash content, and high average smoke concentrations during "off" periods of stoker operation.

As shown in Table 3, there was gen-

eral agreement between conclusions drawn from results of evaluation tests and the marketing experiences of the coal operators. Those coals classed as unsatisfactory have not been prepared by the respective operators in residential stoker sizes. Most of the coals rated as being satisfactory are either being marketed successfully, or will be marketed when preparation facilities become available. Of those rated as not being completely satisfactory, three have been marketed successfully; the others are not being prepared now in residential stoker sizes, or information regarding them is lacking.

The agreement between results of the evaluation tests and marketing information is reasonably good and appears to justify use of the method. Because it is a relatively short-time test, an appraisal of a coal can be accomplished much more quickly than if adequate field tests were attempted. Many of the requests for tests come in summer months when operators are interested in the market for the following heating season.

The method possesses limitations, particularly in regard to the evaluation of clinker formation with coals of intermediate ash-fusion characteristics. With coals of this class having low ash content, the fraction of the ash formed into clinker may be unduly low, although if the test were continued, the fraction would become greater. Admittedly, judgment of clinker formation is less susceptible to evaluation than any other factor involved in the test.

Relation of Results of Test to Standard Laboratory Determinations

Analysis of the data from the evaluation tests has furnished an opportunity for attempts to relate these results to data determined for the coals by standard laboratory determinations. Studies of this kind have been made by Shotts,[2] on Alabama coals, and by Helfinstine and Boley,[3] on Illinois coals. There is a natural interest in correlations of this kind, because of the possibility for simplification of an evaluation test procedure.

INITIAL IGNITION

The variation in the time required for the flue-gas temperature to reach an equilibrium value, as shown by the record of temperature during the first hour of the test, is considered an indication of the performance of the coal with respect to ignitibility and coking characteristics of the coal. Although not a measure of individual characteristics, it gives a preliminary indication of difficulties to be expected in maintaining ignition during operation under hold-fire conditions.

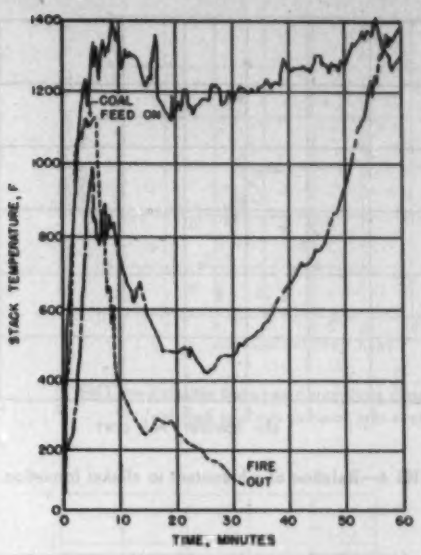

FIG 2—Flue-gas temperature records, initial ignition period.

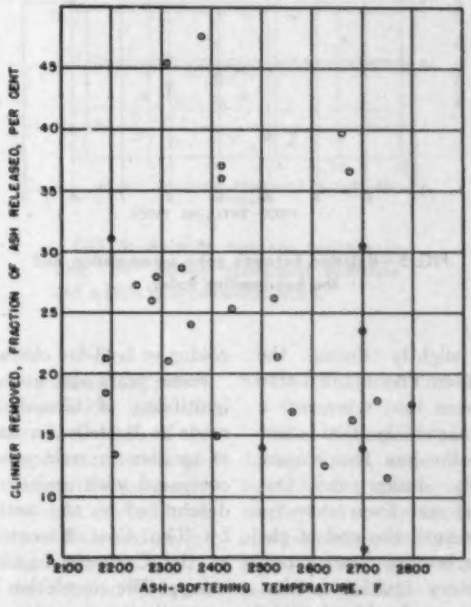

FIG 3—Relation between clinker formation and ash-softening temperature.

Fig 2 presents three examples of initial ignition curves from among the coals tested. These curves emphasize the difference that can exist between bituminous coals in reaching an equilibrium flue-gas temperature value during the initial ignition period. The top curve represents a coal having very good ignition characteristics; the maximum flue-gas temperature was reached early in the hour and

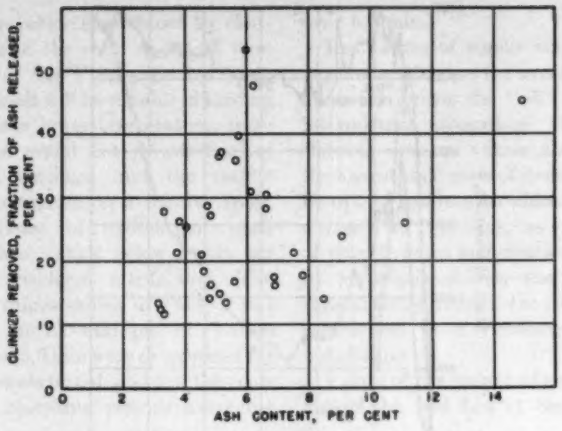

FIG 4—Relation of ash content to clinker formation.

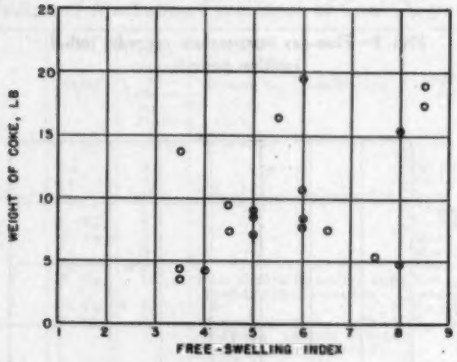

FIG 5—Relation between coke accumulation and the free-swelling index.

declined only slightly during the balance of the hour. The middle curve shown as a broken line, represents a coal having sluggish ignition characteristics; the flue-gas temperature declined sharply during the first 25 min and did not recover to its maximum value until the end of the first hour. The bottom curve represents unsatisfactory ignition in that the fire had gone completely out at the end of 30 min of operation.

An attempt was made to relate data from curves of initial ignition to free-swelling indexes and to the time required for operation of the stoker during periods of hold-fire. This was done by plotting the variance of the flue-gas temperature during the initial ignition period against the free-swelling index, and against the average running time during hold-fire operation. No direct relationship was found to exist between these data; hence, the temperature variation during ignition cannot be used alone as a criterion of coking or hold-fire characteristics.

Some years ago, an investigation of ignitibility of bituminous coals was made at Battelle. In that study, ease of ignition in residential stokers was compared with ignition temperatures determined by the method developed by The Coal Research Laboratory at the Carnegie Institute of Technology. The correlation between these properties was not good. It is concluded that ease of ignition depends not only on ignition temperature, but also on other factors, chiefly coking characteristics and size variation.

CLINKER FORMATION

Results of clinker formation in the evaluation test are expressed as the percentage by weight of the total ash released during the test which is formed into plus 3 in. clinker. If values for these percentages are 20 pct or higher, the coals may be expected to give satisfactory service to the consumer in this respect; below this value, the clinkering characteristics of the coal are open to question.

Comparison of clinker values obtained in these tests and ash-softening temperature shows no correlation. Fig 3 presents a graph of the relation between ash-softening temperature, as determined by the standard laboratory method, and the clinker removed from the fuel bed in the burning tests. Poor correlation is evident between these data. Similar results have been obtained by Helfinstine and Boley.[3]

The ash content of the coal as determined by the proximate analysis definitely predicts the degree of manual attention that will be required when burning the coal, due to the removal of clinker and unfused ash.

Fig 4 presents the relation between the ash content of the coal and the clinker formation. The relationship here is little better than that shown in Fig 3; however, no single laboratory-determined characteristic can accurately predict the performance of a stoker coal with respect to clinker formation.

COKE FORMATION

In the evaluation test, a numerical measure of coke formation is obtained by the weight of plus 1 in. coke remaining in the fuel bed at the end of the test. The standard laboratory test used to obtain an indication of the coking tendencies of a coal is the free-swelling test as prescribed by ASTM designation D-720-46.

Fig 5 shows the relation between these two methods. The correlation is not as close as might be expected, though the results are in agreement with those of Shotts.[2] The reason for this lack of correlation is largely the result of variations in the reactivity of the coke formed.

RATE OF FUEL CONSUMPTION DURING HOLD-FIRE OPERATION

The stoker running time necessary to maintain the fuel bed in good condition is of importance to the consumer for reasons of economy and comfort, as it determines the amount of coal that will be burned during periods when there is no demand for heat.

An investigation was made of the relation between the time required to hold fire and laboratory-determined characteristics. Between time required to hold fire and the free-swelling index no relationship could be determined. A

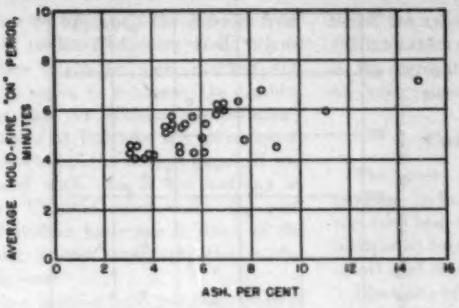

FIG 6—Relation between ash content and average time "on" during hold-fire operation.

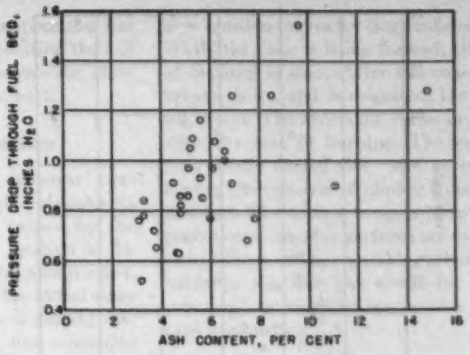

FIG 7—Relation between percentage drop through fuel bed and ash content, per cent.

plot of the relation between hold-fire time and the ash content is shown in Fig 6. These data indicate that, in general, the hold-fire time is increased slightly with increase in the ash content of the coal.

RESISTANCE OF THE FUEL BED TO THE FLOW OF AIR

The resistance of the fuel bed to the flow of air is of interest, as it may reflect the general condition of the fuel bed and the ability of the stoker fan to furnish the necessary volume of air. The fan capacity of residential stokers is sufficient to overcome maximum fuel bed resistances of 1.50 to 1.75 in. of water. With suitable automatic air controls, the delivery of air will be maintained up to this pressure. The highest resistance found during the present study was 1.54, and, for most of the coals tested, values were well below the limiting pressure.

The principal factors contributing to fuel bed resistance are coking properties, size of the coal, and ash content. No correlation was found between free-swelling index, as indicative of coking properties and fuel bed resistance. Fig 7 presents a plot showing the relation of fuel bed resistance to the ash content of the coals. The effect of increasing ash content upon the resistance offered by the fuel bed is evident in this graph.

UNIFORMITY OF BURNING

The ability of a stoker coal to respond to abrupt demands for heat, and to burn uniformly during periods of heat demand are very important factors in providing satisfactory operation for the consumer. The uniformity of burning is judged mainly by the performance of the coal during the period of intermittent operation on the last day of the test. A measure of the performance during this period is obtained by the operating time necessary to liberate a given quantity of heat, and the uniformity of these values for successive operating intervals.

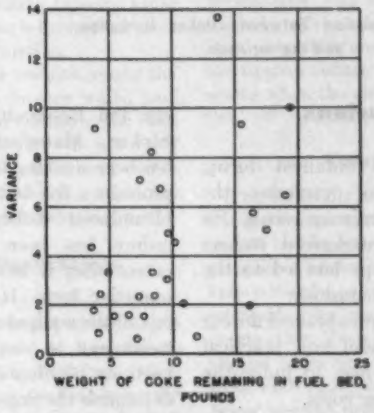

FIG 8—Relation between variance in time "on" during intermittent operation and weight of accumulated coke.

Fig 8 shows the relation between the variance in time "on" during intermittent operation and the weight of coke remaining in the fuel bed at the end of the test. The trend of the data in this graph shows that increased quantities of coke are reflected in increased variations in the uniformity of burning. Fig 9 presents a similar plot for clinker formation and variance. Here it is shown that the amount of clinker formed had much the same effect on the uniformity of burning.

A study of data relating the free-swelling index and the average "on" period during intermittent operation showed a complete lack of correlation.

SMOKE CONCENTRATION

When the stoker is in operation during a test, under conditions of at least 30 pct excess air as normally employed, smoke concentrations are of no consequence, even when burning high-volatile coals. Smoke emission is noticeable only for a brief period after the stoker shuts off.

Of the laboratory-determined characteristics, volatile matter would appear to be the most closely related to smoke concentration; the volatile matter content usually is associated with the relative smokiness of a coal.

Fig 10 shows in graphic form the relation of volatile matter and smoke concentration for the coals tested. In general, the correlation of the data for these two characteristics is poor, indicating that volatile matter alone cannot determine the amount of smoke that will result from burning a coal in residential stokers.

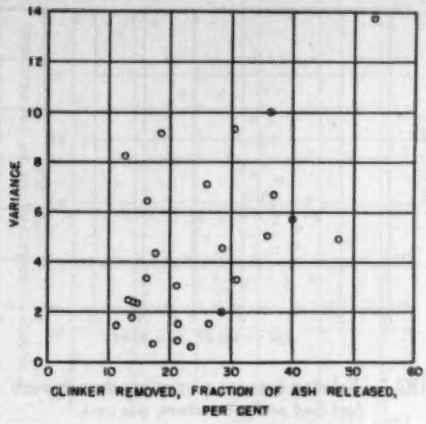

FIG 9—Relation between clinker formation and the variance.

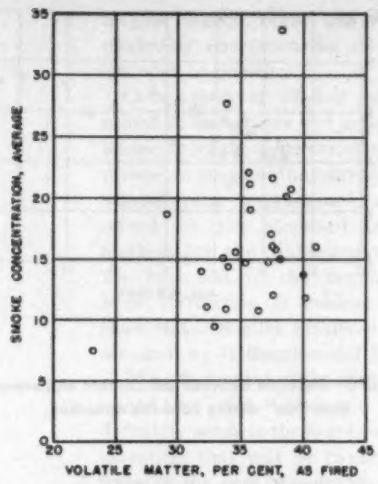

FIG 10—Relation between smoke concentration and volatile matter content.

Conclusions

A study of data obtained during laboratory tests to determine the suitability of bituminous coals for use in residential underfeed stokers of the clinkering type has led to the following general conclusions:

1. Performance data obtained during the test furnish useful and practical information upon which to judge the performance of stoker coals.

2. Conclusions derived from the results of evaluation tests are in general agreement with use experience of the coals in the field. The method is of practical use in this regard.

3. Few direct relationships were found between data furnished by the evaluation test and determinations by standard laboratory tests. This is particularly true with regard to coke formation and clinker formation, two of the most important factors in determining the degree of satisfaction to be obtained in applications of the small underfeed stoker.

It has not been the intention in this paper to prescribe this evaluation test as the best method for the evaluation of coals for use in residential stokers. That there are other test methods having similar objectives serves to point out the necessity for full-scale laboratory tests.

At the present time, a standard test procedure for the evaluation of coals for use in residential stokers is not available. For the past several years a joint committee, appointed from the membership of the Residential Stoker Committee of Bituminous Coal Research, Inc., and from the Engineering and Research Committee of The Stoker Manufacturers Association, has been working to develop a standard procedure for testing and evaluating bituminous stoker coals. This procedure has been completed to the extent that it has been drawn up in tentative form. It is now planned to install the equipment in several laboratories and to run check tests, using portions of identical lots of coal, to determine the degree of reproducibility offered by the method. Completion of this standard test procedure should greatly facilitate the evaluation of coals for use in residential stokers.

Acknowledgments

The cooperation of coal operators sponsoring the coal evaluation tests, in granting permission to use data and results from their tests, is gratefully acknowledged. Acknowledgment is also made to Carroll F. Hardy, Chief Engineer, Appalachian Coals, Inc., for his interest and timely suggestions on the preparation of this paper. Special thanks are due to Ralph Sherman, Assistant Director, Battelle Memorial Institute, who was largely responsible for development of the evaluation procedure, for helpful suggestions and advice.

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DISCUSSION

E. R. Kaiser*—As a former fuel engineer assigned to residential stokers at Battelle, the writer has read with considerable interest the paper by Messrs. Purdy and Nelson. It is gratifying to note that earlier test techniques have been developed and that criteria found important in the years 1935 to 1938 are still important in judging the performance of stoker coals for residential heating.

The authors have plotted a number of the primary coal data singly against performance data of the stoker in an effort to establish relationships. Unfortunately, wide scattering of points usually resulted, which did not illustrate what experience generally indicates to be the case. For example, Fig 3 indicates almost no trend in the percentage of released ash converted to clinker with change in ash-softening temperature. Other factors, such as fuel bed temperatures and zones of heat release must have influenced the results.

In the present state of our knowledge, we cannot explain all of the reasons why suitable stoker coals perform satisfactorily, nor why one satisfactory coal may be better than another. It would therefore

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be better to approach the subject from the side of unsatisfactory coals whose properties might be more marked and therefore easier to evaluate. On examining the data for sixteen unsatisfactory coals tested at Battelle, not all of which were reported in the subject paper, it was concluded with one of the authors as follows: Unsatisfactory coals for residential stokers have one or more of the following general characteristics in the Battelle test:

1. Slow ignition with less than 20 lb of coal burned during the first hour.
2. Over 8 pct ash content.
3. Over 2600°F ash-softening temperature.
4. Less than 20 pct of the ash released converted to clinker.
5. A smoke reading of over 12 units average during the "off" period.
6. A windbox pressure exceeding 1.25 in. w.g.
7. Weight of coke at end of test exceeding 10 lb.
8. Free-swelling index exceeding 8.0.
9. It is assumed that the top size does not exceed 1¼ in. and that the amount of minus ¼ in. coal does not exceed that of normal resultant coals.

Returning to the question of correlating stoker performance with basic laboratory tests, the stoker performance is so complicated that it is not reasonable to expect the performance factors to vary with single coal factors. At least two or three coal characteristics are at work at the same time in affecting clinker formation. These factors are discussed under the same headings as used by the authors.

Ignition Properties

A coal with poor initial ignition shows symptoms of poor ignition later during "on" periods and hold-fire. Coking characteristics and size consist may be factors, but something inherent in the coal other than that measured by the Coal Research Laboratory method seems to be present. Perhaps it is liberation of CO_2 and water vapor from the coal, with absorption of heat, when the coal is subjected to ignition. Weakly coking coals from the Indiana-West Kentucky field sometimes exhibit poor ignition. Outcrop coals are notorious in this respect. Additional basic research is needed on this subject.

Clinker Formation

Clinkering of coal ash is evidently affected not only by the ash-softening temperatures but by the temperatures in the ash zone. A hot fire in the region beginning immediately above the tuyeres and extending at least 5 in. upward will promote clinkering. A coal that ignites slowly and produces coke rather than heat is not likely to promote clinkering. A high ash content does not help rapid ignition nor rapid combustion. Furthermore, the ash-softening temperature has the limitation of not representing the ash in its segregated or heterogeneous state on being released from the coal.

Coke Formation

The poor correlation between free-swelling index and weight of coke in the fuel bed is probably caused by the additional factors of slow ignition in the retort and reactivity of the coke formed.

Plasticity of the coal in the initial stage of coking is probably the dominant factor, with ignition second, and reactivity of the coke third. The plastic property obviously must be first in importance in forming the coke tree. Slow ignition causes green and unoxidized coal to move up into the heat zone where the temperatures are better for coking than for rapid combustion. Once coke is formed, air cannot penetrate the formation.

Because of hold-fire conditions and the cooling effect of the furnace walls, particularly in household furnaces, the coke loses ignition and often will not burn until it falls to the hearth and is reignited by the flames issuing from the annular space above the tuyeres.

Hold-fire Operation

The ignition property of the coal must again be a dominant factor in the length of time the stoker must be operated to brighten the fire. It is therefore understandable why the free-swelling index did not alone prove to be the factor. A small fuel bed of hot coal with a low fuel maintenance is all that is necessary. A heavy fuel bed of smoldering coal results if the stoker feed screw continues to deliver coal that will not ignite at the rate fed.

Fuel Bed Resistance to Air Flow

Coals that ignite slowly often cause high windbox pressures. Green coal floods the tuyeres rather than burning away promptly in the air jets. Low windbox pressures result when the combustion is rapid and a free annular space is left around the coke tree and inside the clinker ring.

Any unclinkered ash that falls into the annular space contributes to the air resistance. The windbox pressure also increases gradually with the amount of clinker in the bed simply by increasing the height of the annular space through which the gases must flow.

Uniformity of Burning

Coals that ignite promptly and do not coke too strongly burn with a small fuel bed. The uniformity of burning is good, neither too much nor too little, but approximately at the rate the coal is fed.

The combination of strong coking and slow ignition makes for large coke masses. While the coke is being formed, the rate of burning is slow. After the coke mass breaks down and is reignited, the burning is fast. The excess air varies inversely with the rate of burning. The extreme case occurs during the "on" period following the removal of clinker from a hot fuel bed. The coke is broken down in the process and reactive surfaces are exposed. Long flames and appreciable carbon monoxide in the flue gas result for a few minutes following the resumption of full rate combustion.

Smoke Concentration

Poor ignition can also contribute to off-period smoke. Smoke increases with the amount of green coal and partially devolatilized fuel above the retort, particularly at the center of the fuel mass. Prompt ignition and intense burning near the tuyeres reduce the coal left to cause smoke when the air flow is shut down for the "off" period.

Conclusion

The foregoing indicates strongly that the ignition property of bituminous coals should be investigated in a basic way to establish a laboratory method that would help not only in evaluating coals but in analyzing the performance of coals in combustion. Tests conducted at Battelle several years ago with fuel beds ignited from above and supplied with air from below showed promise of obtaining useful results.

Further study might well be given to the noncombustible gases liberated from coal on heating above room temperature. The heat required to raise the temperature of the coal, including that for liberating the gases, should also be measured. This heat is not important in relation to the total calorific value of the coal, but in underfeed ignition the amount of heat radiated in the ignition zone is small enough to make this factor highly important. If the tests could be conducted on the same coals used in the laboratory stoker tests it may be possible to develop a better correlation of all of the data obtained thus far, and to place a measured value on the ignition factor.

J. B. PURDY (*author's reply*)—Mr. Kaiser's knowledge of the background of these tests has provided a very interesting discussion. The limiting values generally found to be present among coals not considered suitable for use in residential underfeed stokers of the clinkering type, have been clearly summarized.

Of special interest are the comments, made in Mr. Kaiser's conclusion, regarding the need for additional basic research on the ignition properties of various coals. It is hoped that additional work can be accomplished along these lines in the near future.

Work of the U. S. Geological Survey on Coal and Coal Reserves

By PAUL AVERITT*

The U. S. Geological Survey has been actively engaged in work on coal for more than 50 years. During this long period we have released more than 300 publications containing information about coal and coal reserves, including details on the thickness, distribution, and quality of coal, and the structure and stratigraphy of the coal-bearing rocks. We have also cooperated with state surveys on many similar publications. For many areas these reports are the only reliable published data on our resources of coal.

During the last several years we have substantially increased our activities above the wartime low. We now have field work in progress, or reports in preparation, on 13 detailed mapping projects distributed as follows:

1. Pennsylvania anthracite field
2. Casselman Basin, Md.
3. Leslie County, Ky.
4. Deep River field, N. C.
5. Coosa field, Ala.
6. Powder River field, Mont.
7. Spotted Horse field, Wyo.
8. Yampa field, Colo.
9. Paonia field, Colo.
10. Trinidad field, Colo.
11. Durango field, Colo.
12. Chaco River area, N. M.
13. Lewis and Thurston County fields, Wash.

For two of these areas, the Paonia field, Colo., and the Coosa field, Ala., we have published recently preliminary reports, and final reports are in preparation. In addition, we are nearing completion of a coal reserve study of Montana, and are beginning similar studies in several other states, as part of a program of revising our estimate of national coal resources on the basis of presently available data.

Factors Involved in Making Estimates

Our responsibilities include detailed field mapping to secure information on areas of outcrop of coal beds, their range in thickness, nature of roof rock, amount of overburden, correlation of the beds, stratigraphy and structure of the coal-bearing rocks, and the preparation of coal reserve figures based on these data. The computation and estimation of resources of coal in the ground, like that of resources of other mineral commodities, involves consideration of many additional purely geological factors that can be appraised best through such regional studies. Regional variations in the thickness and texture of rocks overlying and underlying coal beds, for example, give evidence as to the probable position of the edges of the basin of coal deposition, or may aid in the delineation of sand-filled channels that sometimes cut out coal beds. It is apparent that the detailed field studies, in which data are gathered, are the basis for coal reserve studies, and in our present program these two phases of work are being carried on together.

White Sulphur Springs Meeting, November 1948.

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* U. S. Geological Survey, Washington, D. C.

Previous Coal Reserve Estimates

For many years we have been aware

of the need for a substantial revision of the Campbell reserve figures, which were last revised by him in 1928, and which have been used widely since that time with only minor changes. We had little opportunity, or funds, for this work during the years following 1929. However, we did accumulate much new data, particularly on Montana, and when we were able to start a new reserve tabulation, we began our national program with a reappraisal of our accumulated data on reserves in that state, while we stepped up our detailed work elsewhere.

My examination of published statements about coal reserves together with our own experience, has clearly demonstrated the fact that a reserve statement is of little value unless a considerable amount of data about how limits of coal are established at depth, minimum thickness of coal considered, weight of coal assumed, and similar factors are provided to show how the estimate was prepared. I am surprised at the number of estimates that do not give this information in full, and at the number of writers and speakers on the subject who attempt to compare estimates made on entirely different bases. Mr. McKay's new estimate of Canadian reserves, which was made most carefully and conservatively, and which includes only the thicker beds currently considered to be minable, cannot be compared with Mr. Campbell's estimate of the coal in the United States, which includes bituminous coal 14 in. or thicker to a depth of 3000 ft. Nor can an engineer's realistic appraisal of coal reserves in beds 4 ft thick or greater within 2 miles of the outcrop, be compared with more generous estimates of total coal in thinner beds and to greater depths. Yet I have seen the published data abused in this manner.

Considerations for Current Survey

In our current work on coal reserves we have had to face several problems peculiar, more or less, to this country. We felt impelled to maintain the 14 in. cutoff established by Mr. Campbell so that new state estimates would be comparable to the older estimates, and so that our tabulation would be complete. It was necessary, therefore, to establish several intermediate cutoff thicknesses, so that data would be available for beds of present economic interest. For the same reason we wished to maintain 3000 ft as the limit of overburden under which mining can be carried out, though obviously, values for intermediate ranges of overburden would have to be calculated. Finally, we had to evolve systematic methods for handling the vast amount of outcrop data available in certain areas where little mining or drilling has been carried on, and where information about coal at depth is not available. With these few technical restrictions we evolved, with help from Mr. Toenges and Mr. Turnbull of the Bureau of Mines, a series of procedures for calculation and tabulation, which we have been following, and which I should like to present at this time for your consideration and comment. Although systematic procedures, and seemingly arbitrary definitions, are necessary, it should be noted that these are guides only, and must be so used. Actually, geologic interpretation and sound judgment enter into the application of these definitions and procedures to a very considerable extent, as is true of all statements of policy and objectives.

Definition of Terms

First, we adapted the standard Geological Survey-Bureau of Mines nomenclature for reporting reserves of mineral commodities to fit the special case of coal. This permitted us to define three classes of coal reserves according to the reliability of data available for making calculations.

MEASURED COAL

"Measured" coal, we say, is coal for which tonnage is computed from dimensions revealed in outcrops, trenches, mine workings, and drill holes. The points of observation and measurement are so closely spaced, and the thickness and extent of the coal is so well defined that the computed tonnage is judged to be accurate within 20 pct or less of the true tonnage. Although the spacing of the points of observation necessary to demonstrate continuity of coal will vary in different regions according to the habit of the coal beds, we are assuming that for "measured" coal the points of observation will be, in general, on the order of ½ mile apart. The outer limit of a block of "measured" coal, therefore, is of the order of ¼ mile from the last point of positive information (that is, roughly one half the distance between points of observation).

Where no data are available other than measurements along the outcrop, but where the continuity of the outcrop is measured in miles, and suggests the presence of coal at great distances in from the outcrop, we feel that a certain amount of coal should be classified as measured. We are proposing, therefore, a smooth line drawn roughly ½ mile in from the outcrop, be used to mark the limit under cover of a block of coal that can be classed as "measured."

INDICATED COAL

"Indicated" coal is coal for which tonnage is computed partly from specific measurements, and partly from projection of visible data for a reasonable distance on geologic evidence. In general, the points of observation are of the order of 1 mile apart, but may be as much as 1½ miles for beds of known geologic continuity. For example, if drilling on ½ mile centers has proved up a block of "measured" coal of fairly uniform thickness and extent, the area of "measured" coal as defined above is, according to the judgment of the estimator, larger than the actual area of drilling by as much as ¼ mile on all sides. If from geologic evidence the bed is believed to have greater continuity, this area of "measured" coal is surrounded by a belt of "indicated" coal, which according to the judgment of the appraiser, may be as much as 1½ miles wide.

Where there are no data available other than measurements along the outcrops, but where the continuity of the outcrop is measured in miles and suggests the presence of coal at great distances in from the outcrop, we propose that two lines drawn roughly parallel to the outcrop, one ½ mile in from the outcrop, and one 2 miles in from the outcrop, be used to define a block of coal that may be classed as "indicated."

INFERRED COAL

"Inferred" coal is coal for which quantitative estimates are based largely on broad knowledge of the geologic character of the bed, or region, and for which there are few, if

any measurements. The estimates are based on an assumed continuity, for which there is geologic evidence. In general, inferred coal lies outside the limits defined for "measured" and "indicated" coal. Where there are only outcrop data on which to base estimates, "inferred" coal is coal lying more than 2 miles in from the outcrop and within limited areas established as I shall describe in a later paragraph.

PROVED, PROBABLE, AND POSSIBLE

It has been suggested that the terms "proved," "probable," and "possible" are in more general use in the coal industry for reporting coal reserves. As I see it, the two sets of terms are loosely comparable, and those who wish to do so might substitute, with perhaps slight revision of the definitions, the terms "proved," "probable," and "possible." The important thing is that such terms be defined.

Methods of Estimating Extent of Coal Beds

We have had to use several ways to limit the assumed extent of each coal bed in preparing our estimates. Where the continuity of the bed is well established by maps of the outcrop, mine workings, and drill holes as in the Pittsburgh bed, the entire area of known occurrence is taken, even though points of observation are widely spaced. Persistent beds that have been traced around a basin or spur are considered to underlie the area enclosed by the outcrop. Otherwise, the length of outcrop within the thickness limits listed above is considered to establish the presence of coal of all classes in a semicircular area having a radius equal to ½ the length of the outcrop. The total area of coal is considered to extend beyond such a semicircle only if mine working or drill holes so indicate; in which case coal is considered to extend only 1 mile beyond the limits of the positive information. An isolated drill hole too far removed to be incorporated in the area thus defined, is considered to determine an area of coal of all classes extending for a radius of ½ mile around the hole.

ESTIMATING WEIGHT OF COAL RESERVES

Many estimates of coal reserves have been based on assumed weights of coal somewhat lighter than available specific gravity determinations indicate. Assumed values as low as 1500 and 1688 tons per acre foot for bituminous coal have been used, for example, in detailed published statements, whereas the average weight of bituminous coal in the ground is nearer 1800 tons per acre foot. We have established, therefore, the arbitrary policy that where other precise data are not available the following values shall be assigned:

	Short Tons per Acre Foot
Anthracite and semianthracite	2,000
Bituminous coal	1,800
Subbituminous coal	1,770
Lignite	1,750

DETERMINATION OF OVERBURDEN

We are working now to provide in our future coal reserves work, very detailed information about the overburden on each category of coal reserves. (By overburden I mean the thickness of rock overlying the coal, regardless of whether it is above or below drainage.) In our work on Montana, however, where topographic information is not always available, we have had to content ourselves with reporting coal between 0 and 2000 ft. As coal is frequently mined under an overburden of more than 1000 ft, and as very little is mined under an overburden of more than 2000 ft, this seemed a reasonable and convenient cutoff, and I wonder if this limit would not be satisfactory in reporting all coal reserves. In most states, however, it will be possible to calculate reserves between 0 and 1000; 1000 to 2000; and 2000 to 3000 ft overburden, and we are prepared to do so.

THICKNESS CATEGORIES

There is some doubt in my mind as to the most desirable thickness categories for reporting coal reserves. At present, in our work on Montana, we are using the following:

For bituminous coal

1. More than 36 in.
2. 24 to 36 in.
3. 14 to 24 in.

For subbituminous coal and lignite

1. More than 10 ft
2. 5 to 10 ft
3. 2½ to 5 ft

It has been suggested, however, that in the future we use the following for bituminous coal:

1. More than 42 in.
2. 28 to 42 in.
3. 14 to 28 in.

I would be very interested to hear expressions of opinion on this most important point.

CALCULATIONS FOR INDIVIDUAL BEDS AND SMALL UNIT AREAS

Finally, we have established to our own satisfaction that it is desirable in calculating reserves to make calculations for individual beds and for small unit areas. In this way the information is of maximum usefulness, and the calculations can be checked and changed as new data become available. We also feel that the legitimate function of the Geological Survey is to compute the total coal in the ground before mining began. This figure is not subject to change because of economic conditions, or changes in mining practice, and it is the base from which coal "mined, and lost in mining" and "recoverable coal" can be obtained by the application of engineering principles.

Conclusion

The coal fields of the United States are large in all dimensions. They cover roughly 350,000 square miles, or approximately one-ninth of the total area of the United States. The coal-bearing rocks commonly are several thousand feet thick, and, as in West Virginia, contain as many as 117 named and correlated coal beds. The thing that impresses me as I watch the progress of our work, is that an estimate of the coal reserves in this great volume of rock is an expensive and time-consuming job. It requires that data be assembled, analyzed, and then presented in numerous categories. Many insidious errors are inherent in the process. People who talk about a quick estimate to "show the facts" are making the error of thinking that because there is an abundance of information, it is easy to make an estimate. Actually, the reverse is true. The more information, the more work required to assimilate it, and the more detail necessary to present it fairly. The "facts" as I see them are that we need in this country a new, detailed estimate of coal reserves, and that to get it will

require both time and money. Any attempt to secure a quick answer will yield a figure that very likely cannot be substantiated, and certainly will not yield information in the detailed form now desired.

It will be about 10 years before a completely new estimate can be prepared for the coal reserves of the United States at our present rate of progress, though of course, much information will be available in the interim. We would like to present such information as we obtain in the form most acceptable to the industry, and to that end we are working closely with coal resource committees of the AIME, the National Bituminous Coal Advisory Council, and the Bureau of Mines.

DISCUSSION

G. H. CADY*—Mr. Averitt's description of the nature of the coal resources investigations of the United States Geological Survey and the progress of this work is very timely in view of the general interest in the subject of coal resources. The Federal Survey is obviously taking the "long view" with respect to the appraisal of the coal resources as part of the preparation of the general geological map of the country and the attendant determination of the quantity of all mineral and fuel resources. It is apparently for regional surveys of the conventional type that the detailed procedures employed by the Federal Survey are applicable. Even if so it is somewhat unexpected to find the same criteria for evaluation will be applied to the country as a whole.

The work on coal resources appraisal has been proceeding at various rates in a number of coal producing states for various periods. Undoubtedly since there has been no common practice in the method of appraisal the states each have worked independently and independently of the Federal Survey, and each has adopted certain practices to meet local conditions. I do not know how much consideration has been given by the U. S. Geological Survey to these practices. It seems probable that such consideration might result in important modifications of the standards which apparently have been set up by the Federal group in line with local requirements.

It is the practice in some state surveys to present the facts in regard to the occurrence and distribution of coal beds and their variations in thickness down to thicknesses of 1 to 1½ ft in relatively narrow stages up to maximum thickness. It is then possible for any one using such figures to compile quantitative estimates of workable coal at various minimum thickness limits. This is a desirable objective where surveys are detailed and information correspondingly good. However, vast quantities of coal lie in beds that are relatively thin about which little information is available and the quantity of such coal might affect the final appraisal to the extent of several thousand million, depending upon the minimum limitation observed. Thus in the case of Illinois a difference of 1 ft in average thickness of coal in the coal field would make a difference of approximately 35 billion tons in the estimated quantity present. One is dealing with a very large area and small differences in thicknesses are very important in the overall picture.

The estimate of 10 years to accomplish the sort of detailed appraisal outlined by Mr. Averitt is viewed somewhat skeptically and certainly would require many more trained geologists than seem to be available. The requisite detailed mapping and study are very time consuming not only because of the geological field work required, which is difficult in certain seasons, but also in order to assemble data from company and personal files, and to study drill records and cores, and to sample and analyze the coal. Unless the staff of geologists could be increased in the order of several hundred per cent the prospects are that the appraisal of the type suggested will require many decades not just one. The permanent value of such appraisal, however, is not questioned and it should be expedited as rapidly as means allow.

Satisfactory appraisal is not easily achieved and will vary from time to time. The main function of a geological survey is to collect and present the facts that will make possible appraisal in terms of existing conditions by individuals, particularly engineers, qualified by training and experience for making such estimates. Thickness and character of the coal bed, nature of the overlying and underlying strata, position, thickness, and character of "partings," relationship to other coal beds, and a variety of other factors must be assembled by the geologist in order to present a complete picture of the geological setting. The economic and engineering factors are in general outside the field of geological experience. The geologist should see to it that the pertinent geological facts are available; much more information than simply thickness and depth of the coal beds is essential to meet all the requirements of appraisal in the future as well as at present.

With respect to the present demand by engineers and coal mine operators and others for a rapid re-appraisal of the available coal resources, in the light of existing practices and those in immediate prospect, the Federal procedure of systematic mapping and appraisal is scarcely in line. It will be necessary for those particularly interested in having such an estimate made to establish standards with respect to the various factors involved so that the scope of the geological work can be definitely restricted for the various coal fields. There is little use in setting up general standards since no such standards will be applicable to all fields. Thus it is folly to establish a standard minimum thickness to be generally applicable which is one-half to one-third the minimum minable thickness in some fields. The greatest difficulty in appraisal of the presently actually workable coal bed lies in the variation in the importance placed on roof, floor, structural conditions, bedded impurities, sulphur content and other factors that affect judgment relative to the value of a coal bed. It is doubtful whether any appraisal will have universal approval in view of the variable factors involved.

For this reason if none other, the facts should be carefully and painstakingly assembled after the manner of a systematic geological survey such as that advocated by Mr. Averitt. Only upon the basis of such information can reliable estimates be made in line with selected conditions.

PAUL AVERITT (*author's reply*)—In this paper I discussed, but did not define as such, two major phases of our activities on coal, which are closely related, and which normally we in the U. S. Geological Survey see as an integrated whole. It is understandable, therefore, that Dr. Cady should assume that my remarks applied only to the current work on coal reserves with which I am directly concerned. In view of Dr. Cady's discussion, however, I think a brief summary of the two phases of U. S. Geological Survey activities on coal will be appropriate.

The long-term function of the U. S. Geological Survey on coal investigations is to make detailed geologic studies, involving the preparation of detailed geologic maps showing the outcrops and correlations of coal beds, direction and dip of coal-bearing rocks, nature and thickness of overburden, location of faults and fold axes, and similar features. Such studies also yield many measured sections of coal and associated strata, which show variations in the thickness of the beds, and the intervals between coal beds. Taken as a whole, detailed geologic studies provide much useful information for mine operators and engineers. As Dr. Cady points out, this type of geologic work is arduous and time consuming, but it provides the basic data that are necessary both in the development of our coal fields, and also in the preparation of coal-reserve estimates, which are discussed in the next paragraph. The U. S.

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Geological Survey and the state surveys have been doing such detailed work for more than 50 years, yet some areas in the United States have never been mapped, and in many others the work is only of a reconnaissance nature. Dr. Cady is correct in his statements to the effect that the detailed mapping to modern standards of all the coal fields in the United States would require a large staff of geologists for many decades. This, then, may be regarded as the continuous, long-term phase of our activities, which has been in progress for more than 50 years, and may still be incomplete 50 years hence.

From time to time in the course of this long-term program of detailed field surveys in the coal-field areas, we have paused to reappraise the coal reserves of the United States on the basis of the available data. Such appraisals were made in 1913, 1928, 1937, and 1943. The interval between 1928 and the present, however, has witnessed both a major depression and a major war, and during this period we were unable to devote the proper amount of time to the revision of the estimates: the estimates made since 1928, therefore, are dependent in large part on old data. We are now engaged in a new appraisal of coal reserves, based on data currently available, and augmented locally by data from our active program of detailed geologic studies mentioned above. This new estimate, which is intended to be somewhat more detailed and thorough than previous estimates, will nevertheless still be only a provisional estimate, and will include much coal that can be classed only as "inferred" or "possible" because of the paucity of information. Inevitably, such an estimate, like all reserve estimates, will be subject to modification the day after it is prepared. I am sure Dr. Cady will agree that a new estimate of reserves can be prepared on this basis in a modest length of time, depending, of course, on the size of the staff so employed.

Dr. Cady's comment about the possible value to the U. S. Geological Survey of procedures employed by the several state surveys that have estimated their reserves is, of course, most appropriate. I examined with profit the procedures employed by the Pennsylvania, West Virginia, Ohio, Illinois, Kansas, and Canadian surveys, before formulating our own. I also submitted our method of procedure to the Committee on World Coal Resources of the AIME, of which Dr. Cady is a member, and received several very helpful suggestions, which I here take opportunity to acknowledge. Of the procedures employed by other geological agencies, however, only the Kansas and Canadian surveys attempted a breakdown of the data by classes according to the reliability of the information, and none have published a breakdown by range of thickness of beds, though all have provided much data on the range of thickness and average thickness of the individual beds. I believe, therefore, that having profited by our own past experience in estimating reserves, and having borrowed what we could from our friends, we are on the way toward a new reserve figure that will be an improvement on previous estimates, and will present data in the detail required for modern needs. It is our hope, of course, that all state surveys will collaborate on coal-reserve estimates to some standard specifications. Some will doubtless be able to contribute more in data and evaluation than others, but all should participate. Naturally we expect to arrive at mutual understandings with the state surveys regarding the scope of responsibilities to be assumed.

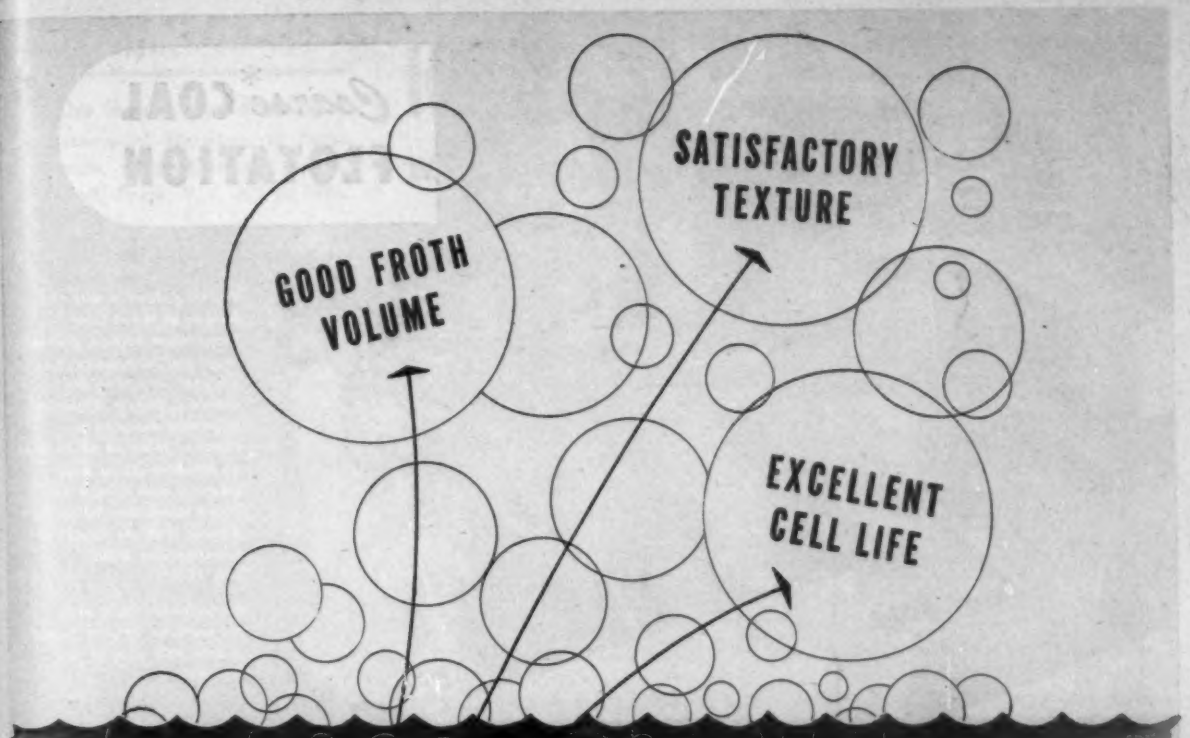
I agree with Dr. Cady that the main function of a geological survey is to collect and present the facts, because the fundamental basis for all coal-reserve estimates is furnished by geologic surveying of coal outcrops, the establishment of the correlation of the coal beds between points of observation, and the measurement of bed thicknesses. It is likewise a function of the geologist to estimate the reserves of coal in the ground, which calculations can be made directly from the geologic data, and other data obtained during the course of field surveys. Naturally, studies of recoverability and minability are the functions of mining engineers. It is, however, the duty of geologists to present their data on reserves in the form that will be most useful to aid in studies of minability, and to aid those who would mine the coal, or make plans based upon its minability. It is for this reason that we are now making reserve estimates in several classes, both according to the reliability of the data on which the estimate is based, and according to the range of thickness of the beds.

The need for a breakdown of coal-reserve data into classes will materially increase the time required to do the job, but this is an inevitable consequence of the demand for more detailed information. A better job always takes longer, or takes more men. The U. S. Geological Survey can complete the coal-reserve appraisal in less time if larger funds are made available.

The establishment of standards and procedures for carrying on coal reserves work is one of the most difficult problems to resolve. As Dr. Cady points out, "In the case of Illinois, a difference of 1 ft in the average thickness of coal in the coal field would make a difference of approximately 35 billion tons in the estimated quantity present." It is obvious that coal reserves no longer can be estimated on so gross a scale. It is obvious also, that some procedures must be established for carrying on the work. Some simple, reasonable procedure is required, for example, in weighting the average thickness of coal used in making the estimate for each individual bed. Likewise, there must be some simple, reasonable procedure for defining the area of coal to which a given thickness is applicable. Otherwise, large gross errors of overestimation will be perpetrated.

The principles we have established have evolved slowly as new problems have arisen and new suggestions have been received. We have, for example, accepted the recommendation of the National Bituminous Coal Advisory Council that reserves be calculated for bituminous coal in the thickness range between 28 and 42 in., and if possible, at 1 ft intervals above 42 in. We have also accepted the recommendation that the reserves be calculated for the several subdivisions of the bituminous rank, and, wherever possible, for sulphur content in certain ranges. Dr. Cady's statement that "There is little use in setting up general standards . . . " reflects, I think, his long experience in coal reserves work, and his awareness of the magnitude of the problem of calculating reserves to fit every specification and every set of mining conditions. We, too, are aware of this difficulty. The problem should be faced, however, not by abandoning standards and procedures, but by establishing a very few simple standards and procedures that seem to fit most needs, and apply to most regions. This we are trying to do. In practice, the standards and procedures that we have evolved, and that are still changing, are used merely as a general guide to establish a reasonable uniformity of thinking on the part of the geologists engaged in estimating reserves. It is clearly necessary for each geologist to rely heavily on his own judgment, based on his knowledge of the habit of the coal and associated rocks in the area being appraised. As I have stated, "Although systematic procedures and seemingly arbitrary definitions are necessary, it should be noted that these are guides only, and must be so used. Actually geologic interpretation and sound judgment enter into the application of these definitions and procedures to a very considerable extent, as is true of all statements of policy and objectives."

The current interest in coal as a potential source of liquid fuels, and the insistent demand for precise quantitative information about our coal reserves should be very gratifying to many members of the coal industry who have pointed out the need for this very thing for so many years. I think now that with the support of industry and with the application of considerable labor the U. S. Geological Survey and the state surveys acting together can produce during the next few years a new estimate of coal reserves that will serve current needs.



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Coarse* COAL FLOTATION

with DENVER "SUB-A" FLOTATION MACHINES

Working directly with the Bureau of Mines Coal Preparation Section, and with large coal companies engaged in solving the problem, Denver Flotation Engineers designed a Flotation Machine which does an effective job of coal cleaning as will be noted in the tabulations below from Bureau of Mines Report of Investigations 4306, July 1948.

Marketable results are obtained on varying types, grades and sizes of coal fractions.

PROBLEM • Treating coal fractions from 1/16"-0. Two objectives are sought—

1) *to save marketable grade of coal in these sizes and*

2) *eliminate stream pollution from wasted slurry water.*

These coal fractions cannot be satisfactorily cleaned by "gravity" methods. With flotation remaining as the most acceptable solution, a prime consideration is that the flotation machine used must be capable of handling both coarse and fine sizes effectively. The rapid rising, dense agglomerate type "matte" coal concentrate must be removed and dewatered in large volume.

SOLUTION • No commercial flotation machine can treat both coarse and fine sizes so effectively as the Denver "Sub-A." This selective feature plus its extreme mechanical and metallurgical flexibility makes it the logical choice for coal cleaning work.

Removal and dewatering of the dense agglomerated "matte" concentrate was accomplished with special "deep-digging" punched plate rakes which carry the 7"-8" coal matte up the spitzkasten allowing surplus water to drain back in the pulp through the perforated rakes. Further dewatering is carried out in a wedge bar compression screw conveyor. Surplus water returns via middling feed pipe to the Denver "Sub-A" Flotation cell for re-use, thus eliminating surplus slurry water.

	SCREEN SIZE, MESH	FEED		CLEANED COAL		REFUSE		DISTRIBUTION, %		Reagents, lbs. per ton feed
		Weight, percent	Ash, 1* percent	Weight, percent	Ash, 1* percent	Weight, percent	Ash, 1* percent	Cleaned Coal	Refuse	
MINE I *	10 to 14	18.8	11.9	23.0	6.2	9.1	31.6	77.6	22.4	Kerosene, 3.27
	14 to 35	51.4	19.1	56.0	7.6	31.9	65.3	80.1	19.9	B-23 Frother, 0.24
	35 to 100	23.3	40.3	17.0	13.4	35.6	81.8	60.7	39.3	Pine Oil, 0.11
	Through 100	6.5	58.3	4.0	19.3	23.4	84.0	39.7	60.3	
	Composite through 10	100.0	25.2	100.0	8.7	100.0	72.5	74.1	25.9	
MINE II *	10 to 35	64.2	10.5	58.2	3.4	54.0	50.7	85.0	15.0	Kerosene, 3.46
	35 to 100	22.6	10.5	26.9	4.6	22.2	66.2	90.4	9.6	B-23 Frother, 0.26
	Through 100	13.2	12.8	14.9	6.3	23.8	64.8	88.9	11.1	Pine Oil, 0.20
	Composite through 10	100.0	10.8	100.0	4.2	100.0	57.5	87.6	12.4	

1*—Moisture free basis

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Book Reviews

Ore Genesis Hypothesis

Ore Genesis—A Metallurgical Interpretation. By John Stafford Brown. Hopewell Press, N. J. 204 p. 1948. $3.50.

REVIEWED BY W. H. NEWHOUSE

This is a refreshing book. In compact, well organized form the author presents a working hypothesis "that ores were vaporized, chiefly as simple mineral substances from horizons deep within the earth's crust at which they had collected by various processes during the fusion of the encompassing rock materials."

The time sequence of the minerals in ores is believed to be related directly to their specific gravity. Volatilization of the earlier lighter ore materials takes place first from the upper layers of a gravity stratified reservoir, while the heavier and later ore elements come from the lower layers.

Zoning of ores relative to an igneous focus is explained by the relative volatility of the minerals.

The reviewer thinks that some parts of the hypotheses fit well with known facts; the same can be said of the hydrothermal hypotheses. Neither group gives a complete explanation.

The individual ore layers arranged by gravity are hard to accept. Much information that appears trustworthy is hard to reconcile with that hypothesis. Why so many little ore bodies from such a large source? Why no epigenetic ore bodies or districts with billions of tons of ore? The volatility of the sulphides is considered, but what about the relative volatility of the rocks and minerals which they frequently replace? It should be remarked that the analogous data are lacking to support a hydrothermal hypothesis or any other kind of hypothesis. Little work is being done in this important field; instead for several decades, geologists have largely concentrated their efforts on those aspects that might directly contribute to finding ore. As a long term policy this is as sensible as milking the cow but not feeding her. Dr. Brown is to be congratulated for a stimulating book in a field that has become stuffy and dogma ridden, and the St. Joseph Lead Co. is commended for its liberal policy.

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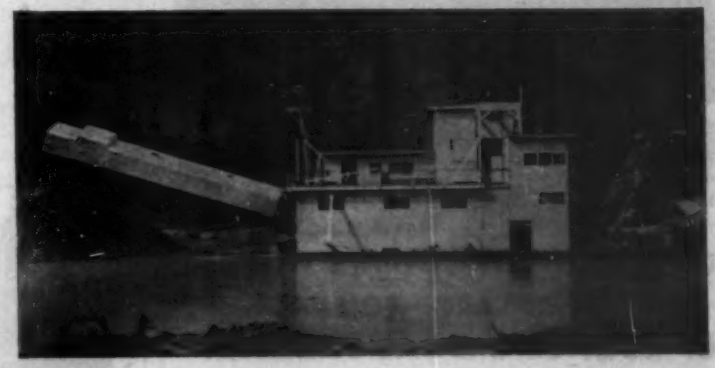

Yuba pioneered the use of alloy steels and heat-treating in the manufacture of dredge parts. You are sure of long-wearing, tough parts when you buy Yuba bucket pins, Abrasion Resisting and Manganese Steel screen plates, tumblers, ladder rollers, pumps, and other dredge parts. Yuba parts are carefully machined, enabling you to make field replacements quickly, and keep your shut-down time to the minimum.

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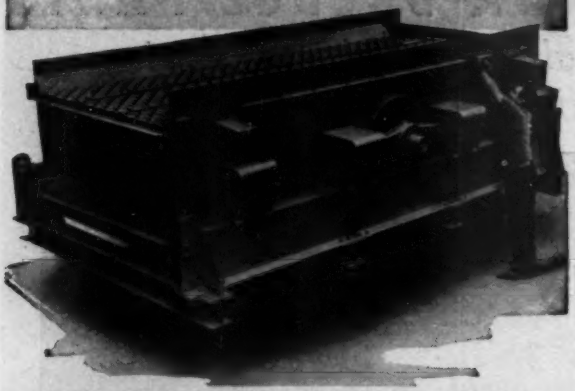

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New Books Received

(*Continued from p. 39, Sect. 1*)

The Gold Placers of Montana. By Charles J. Lyden. Received in Jan. $1. Detailed data on placer deposits and operations in Montana, with 28 detailed maps.

Minerals and Mineral Deposits. By W. R. Jones and David Williams. Received in Feb. $2. Nontechnical history, shape, and internal structure of minerals, and their physical characters, mode of occurrence, and importance.

Patent Law. By C. H. Biesterfield. Received Feb. 28. $4. Presents basic principles of the substantive patent law in understandable terms.

Radioactive Measurements with Nuclear Emulsions. By Herman Yagoda. Received April 7. $5. Coordinated study on the use of photographic emulsions in measuring radioactivity.

Sedimentary Rocks. By F. J. Pettijohn. Received Jan. 28. $7.50. Useful single-volume guide to the observation, classification, and interpretation of sedimentation and sedimentary rocks.

Examination and Valuation of Mineral Property. Third Edition. By Roland D. Parks and W. L. Whitehead. Received in Jan. $5. Aid in the accurate appraisal of mining properties of value to mining engineers, geologists, and oil men.

Engineering the New Age. By John J. O'Neill. Received Feb. 21. $3.50. A challenge to mankind to adopt simple scientific truths in building the world of tomorrow.

International Labor Organization, Coal Mines Committee, General Report. Received April 1. $1.25. The ILO group's report covers their work performed from December, 1948, to this April; an international survey of labor-management relationships in coal mining; and a survey of trends since 1945 within the industry.

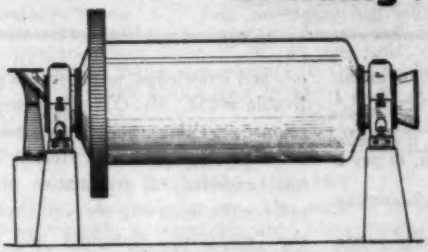
NEW TECHNICAL BOOKS
Engineering Societies Library

New books can be borrowed by AIME members in the United States or Canada for a nominal fee. Bibliographies, photostats, microfilm, and translations can also be supplied.

Advanced Surveying and Mapping. By G. D. Whitmore. International Textbook Co., Scranton, Pa., 1949. 619 p., illus., diagrs., charts, tables, 8 x 5 in., paper, $5.

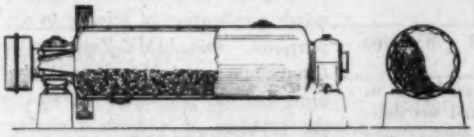
This book provides instruction material for a college course covering such subjects as geodetic surveying and mapping, precise surveying in city work, and aerial photogrammetry. Complete detailed descriptions are given for methods of making field measurements and observations and procedures for performing calculations. A working knowledge of elementary algebra and plane trigonometry is assumed. A brief treatment of the practical application of the theory of least squares is appended.

For Open Circuit Grinding . . .



Hardinge Rod Mill

Minimum oversize. Wet or dry grinding. Convex heads. Self-aligning rods. Bulletin 25-B-2.



Hardinge Tube Mill

For finished product in open circuit. Grinding media — balls, pebbles, or product itself. Bulletin 18-A-2.

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E. J. LONGYEAR COMPANY

MINNEAPOLIS, MINNESOTA, U. S. A.

Gold Mining in South Africa. By C. W. Biccard Jeppe. Todd Reference Library, Todd Publishing Group Ltd., London, W.1. 1948. 160 p., tables, 7½ x 5 in., cloth, 7s.6d.

This book covers the historical, geological, technical and sociological aspects of gold mining in South Africa. It also contains a review of the existing ore reserves and the part the industry must play in the future economy of the Dominion.

Statistical Year-Book of the World Power Conference, No. 4. Data on Resources and Annual Statistics for 1936-1946, edited by F. Brown. World Power Conference, 201-2 Grand Buildings, Trafalgar Square, London, W.C.2. 1948. 212 p., tables, 11 x 8½ in., cloth, £2:5s.

This compilation contains statistics of the resources, production, stocks, imports, exports, and consumption of power and power sources in all the countries of the world for which it was possible to obtain information. The power sources included are coals, brown coal and lignite, peat, coke, manufactured fuel, wood, petroleum, benzoles, alcohols, natural gas, manufactured gas, water power, and electricity. Most of the statistics were supplied by government organizations in the countries concerned, and conform to standard definitions which are reproduced in the text.

Pumping Study

Centrifugal and Axial Flow Pumps. By A. J. Stepanoff. John Wiley & Sons, New York City. 1948. 428 p. $7.50.

REVIEWED BY ARTHUR I. HOEM

This book is a masterly presentation of the theory, design, and application of centrifugal and axial flow pumps, and is the culmination of years of study, research, and experimentation in all phases of the subject.

Dr. Stepanoff, for the past twenty years, collected from all parts of the world nearly all available material on the subject of centrifugal pumps, and the study and use of this material, coupled with theoretical and experimental studies, enabled him to write an authoritative and complete book-length treatise on the art which is commensurate with new developments in the practices and uses of centrifugal and axial flow pumping machinery.

A new method for the theoretical treatment of the centrifugal pump impeller is presented, as is also a resulting simple diagram which combines the important design elements and performance characteristics of centrifugal and axial flow pumps.

Although the book is primarily intended for engineers, consultants, and plant designers in the pump industry as well as for advanced students, preparing to enter the field, it can be read with profit by anyone interested in pumping problems.

It is unfortunate that the outward appearance of the centrifugal pumps is not more pretentious, considering the duty it performs and is expected to perform. Too often the installation and maintenance are also anything but pretentious and the pump suffers accordingly.

Dr. Stepanoff's book tells an unusual story of what takes place inside the relatively drab exterior of the centrifugal pump which inspires respects for this glamourless type of machinery.

(*Continued on p. 42*)

Engineering Societies Personnel Service, Inc.

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In applying for positions advertised by the Service, the applicant agrees, if actually placed in a position through the Service as a result of these advertisements, to pay a placement fee, established to maintain an efficient, nonprofit service.

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All replies should be addressed to the key numbers indicated and mailed to the New York office.

A weekly bulletin of engineering positions open is available to members of the co-operating societies at a subscription of $3.50 per quarter or $12 per annum, payable in advance.

MEN AVAILABLE

ENGINEER AND EDUCATOR, young, married, several years' experience in mineral industries, research, teaching. Doctorate and graduate of several well-known universities. Desires position in academic field. Deanship or chairmanship of department of metallurgy preferred. Some teaching duties with administrative functions desired. M-431.

GRADUATE MINING ENGINEER AND GEOLOGIST, 39, married. Experience in geologic exploration, mine production, examination and evaluation, milling, western U. S., eastern Canada, Philippines. Spanish spoken. Mexico, Central America preferred. Available immediately. M-432.

POSITIONS OPEN

MINE SUPERINTENDENT with experience in cut and fill stoping, mine development, shaft sinking, maintenance. Must have fair working knowledge of Spanish and ability to handle labor. Approximately $9600. Reply giving age, experience, etc. Latin America. Y1320.

ASSOCIATE PROFESSOR, Ph.D. in optical mineralogy and ceramic petrography and some practical experience in petrography and ceramic petrography. $4500 for nine months. Missouri. Y1497C.

MINING ENGINEER, young, two to five years' experience, and with good knowledge of mechanized coal mining. Will function as assistant to executive in New York office and as liaison between mines and office. Salary open. New York City. Y1943.

MINE SUPERINTENDENT, graduate mining engineer, executive type, ten years' experience, four years of which must have been in coal mining; other experience in metal mining will be acceptable where coal mining methods have been used. Will be responsible for laying out plans, developing mechanization methods, etc. $7500. Wyoming. Y1949S.

CHEMIST-ASSAYER, graduate, experienced in fire assaying and wet determinations of lead, zinc, copper, gold, silver, tin, tungsten, bismuth, antimony ores and concentrates, to take charge of laboratory employing twenty workmen and handling about 9000 assays monthly. Standard three-year contract. Working knowledge of Spanish essential. $4200, plus one month's bonus; single status preferred; if married single status for six months, free transportation to Bolivia by air for employee and wife, free living quarters, four weeks' vacation yearly. Y2344.

MINING ENGINEER, recent graduate, for gold mining operation. $2400, plus room and board. Colombia, S. A. Company also requires dredging superintendent. Y2349(b).

MINING ENGINEER with experience in design, underground operations and map work. Salary open. West Virginia. Y2390.

MINING GEOLOGISTS with good technical education and at least two or three years' experience, preferably with mining companies. Salary to start, about $3900 depending on experience. Mexico. Y2408.

Book Reviews

(*Continued from p. 41*)

Engineers—High Priests of the Atomic Age

Engineering the New Age. By John J. O'Neill. Ives Washburn, Inc., New York. 1949. 320 p. $3.50.

Man, as engineer, has extracted many of Nature's closely held secrets of the order and harmony which exist in the universe. The engineer, as man, must now do an engineering job on the "new age," combining social consciousness with his technical methods and knowledge, to produce a more livable world. Mr. O'Neill, pioneer in science writing and now Science Editor of the New York Herald Tribune, expounds a provocative philosophy, calls upon engineers to abandon the "executive" viewpoint and start finding "the solution to human problems."

"Engineering is the art of the economic application of science to social purposes," said AIME Past President Gano Dunn, and Mr. O'Neill quotes him. But he realistically delves into the problem of underpaid engineers and those who "must be guided into an engineering career with some assurance," so that they may serve the atomic age as they have served Rome's emperors and the Pharoah's of Egypt—as high priests, into whose hands men willingly surrender their destinies.

Library Service Bureau Rates

A study of library operating costs has shown the need for increasing the rates on some of the services of the Engineering Societies Library to make them self-sustaining. The Library Board has, therefore, authorized the following rates, effective June 1, 1948:

Literature searching (per hour)	$5.00
Translations (per hundred words):	
German, French, Italian, Spanish	$1.50
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Book loans (per week for each volume)	$0.60

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MANUFACTURERS NEWS

Richard W. Berg has been appointed district manager of the Pittsburgh office of *The Torrington Co.* He will coordinate the company's representation to the rolling mill industry throughout the United States.

John Z. Linsenmeyer has been appointed manager of mining, petroleum and chemical engineering for the *Westinghouse Electric Corp.*, succeeding *Phelan McShane*. Mr. McShane has been appointed consulting mining engineer.

Broderick & Bascom Rope Co., St. Louis, has announced two changes in executive responsibility. *Fred Zimmerman*, Vice-President, who has been in charge of sales, will be director of sales, and *J. J. Sieber* has been appointed sales manager.

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Write for Bulletin No. 4710

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(*Continued from p. 42*)

Text for Petrologists

Eruptive Rocks. 3rd Ed. By S. James Shand. John Wiley & Sons, New York. 1947. $7.50. 488 p.

REVIEWED BY T. P. THAYER

Professor Shand has succeeded admirably in preparing a readable, well organized text for the advanced student and for reference by practicing petrologists. The treatment of the principal rock-forming minerals in the light of modern concept of ionic substitution, and the chapter on fugitive constituents of magmas, serve as a good foundation for the discussions of magmatic history — assimilation, crystallization and differentiation — that constitute about half the book. Shand's emphasis on assimilation in the genesis of various magma types should help the student to evaluate the problems of petrogenesis which Bowen explains mainly by crystal differentiation.

The problems of fugitive constituents—"mineralizers"—in magmas are well correlated with late-stage magmatic effects and deposition of certain types of ores. The author fully recognizes our present ignorance regarding the origin of sulphide ores. It is somewhat surprising, after arguments pointing to a very low hydrous content in most magmas, for the author to accept serpentine magma apparently without question!

Discussion of rock classification is limited to two chapters, and rock descriptions to four chapters at the end of the book. All students should appreciate Professor Shand's efforts to keep rock names simple and descriptive, even though they may not use his proposed system of symbols.

Though some of the illustrations leave much to be desired, the bibliographies at the ends of the chapters are very good. Professor Shand's volume merits careful consideration for a place on the bookshelf of every geologist at all interested in "hard rocks."

(*Continued on p. 46*)

with the
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Now—a revolutionary new radioactivity detector—the BECKMAN "MX-8"—greatly simplifies modern uranium prospecting. Developed by one of nation's leading manufacturers of scientific instruments, the "MX-8" combines high sensitivity with rugged design and featherweight compactness—*at a price that places uranium prospecting within reach of everyone!*

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3 FLOAT-SINK CONCENTRATOR—Link-Belt — This new concentrator has achieved unique success in coal cleaning and can be adapted to the concentration of other nonmetallic or metallic ores that are amenable to separation by gravity methods. Bulletin 2101.

4 MOLY CAST IRON — Climax Molybdenum Co.—Some high pressure steam valves are made of cast iron containing 3.10% Total Carbon, 2.45% Silicon, 0.75% Manganese with 0.75% Molybdenum added. This composition, and hundreds of other examples of the economical application of Molybdenum cast iron, are to be found in a new booklet, "Applications of Molybdenum Cast Irons."

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10 TRACTORS—Caterpillar Tractor Co.—"Clearing for Action," a 15-page illustrated booklet, describes clearing methods for reclaiming productive agricultural land, construction sites, mine and quarry operations, and housing projects. Form 11796.

11 ROTARY POSITIVE BLOWERS—Roots-Connersville Blower Corp.—A new Bulletin 22-23-B-13, on rotary positive blowers, outlines in its foreword some of the principal uses for these blowers which had their inception in 1854 by the Roots brothers.

12 EXCAVATOR — Harnischfeger Corp.—Copy of a new bulletin, X71-2, covering the P&H Model 255A ¾ yard excavator is now ready for distribution. It embraces 24 pages, is printed in 2 colors, and contains many photographs of the machine equipped as a shovel, dragline, clamshell, crane, trench hoe, and pile driver.

13 MAGNETIC PULLEYS—Stearns Magnetic Mfg. Co.—Bulletin 350 describes a new line of permanent magnetic pulleys. The pulleys are nonelectric, require no wiring or electrical connections, and can be used under all atmospheric and operating conditions for automatic removal of tramp iron, reclamation of secondary metals, and for separating magnetic and non-magnetic materials.

14 CEMENT GUN — Cement Gun Co., Inc.—Of considerable interest to the engineering profession is a 65-page booklet, designated as bulletin 2400, describing the cement gun and its applications. It covers the principal uses of "Gunite" and is profusely illustrated. In the last few pages of this bulletin are typical specifications for various types of "Gunite" work.

15 DIESEL ENGINES AND POWER UNITS—International Harvester Co.—A 24-page catalog covers 6 diesel engines ranging from 39 to 180 hp. Design features are shown through many sectional views and performance charts reveal brake hp, torque, and fuel consumption at various engine speeds. Specifications, dimension data, and a list of attachments are also given.

16 AUTOMATIC LINE OILER—Gardner-Denver Co.—Information is available on a line oiler that protects against operating rock drills or other air-actuated equipment, without adequate lubrication. Known as the L012 automatic line oiler, this device shuts off the line air automatically when all the oil in the reservoir has been used.

17 ALLOYED IRON AND STEEL—The International Nickel Co., Inc.—"Notes on Alloyed Iron and Steel" is the title of a new 40-page booklet. It is a collection of selected "case histories" on problems involving materials of construction which were originally published as Questions and Answers in Nickel Steel Topics or Nickel Cast Iron News.

18 AERIAL MAPPING—Aero Service Corp.—Offers catalogs or literature covering its varied aerial mapping services. These include aerial photography,

topographic and planimetric maps from an aerial photographic base, precise aerial mosaics, airborne magnetometer surveys for ore and oil, and both plastic and plaster relief maps. Services discussed are used in highway design, plant engineering, industrial development, community planning, geological explorations and prospecting for oil or minerals.

19 CENTRIFUGAL PUMPS—A. R. Wilfley & Sons—Many new mechanical improvements, developed over a period of years, are now incorporated in our new models. Interchangeable rubber parts available. An economical pump-size for every purpose.

20 GRINDING, CONCENTRATING, CLASSIFYING EQUIPMENT—Hardinge Co.—Bulletin 100-2 describes a complete line of mining equipment.

21 HYDROSEAL PUMPS — Allen-Sherman-Hoff—Many companies repeat orders for Hydroseal Pumps. Hydrosealing saves 1/3 to 1/2 in power costs and with Maximix Rubber Pumping Parts, maintenance is reduced 4 to 6 times. These pumps are described in Catalog 547.

22 ORE BENEFICIATION EQUIPMENT—Allis-Chalmers Mfg.—Many of the iron ore preparation plants already in operation use Allis-Chalmers equipment to increase production. As 3 tons of low-grade ores have to be processed to produce one ton of concentrate, producers must step up the scale of mining to make low grades pay off. And that's where Allis-Chalmers can help. Reprints describing this equipment are available.

23 PORTABLE SURVEY INSTRUMENTS—El-Tronics, Inc. — Designed specifically for field use in measuring radiation intensities from radioactive elements, the El-Tronics SM3 radiation survey instrument combines rugged lightweight construction with the high sensitivity of delicate laboratory equipment.

24 "MINERAL DRESSING NOTES"—American Cyanamid—Part of Cyanamid's service to metallurgy is "Mineral Dressing Notes," a publication issued whenever new technical data on reagents and processes becomes available.

25 FLEXCO HD BELT FASTENERS—Flexible Steel Lacing—Bulletin F-100 shows how to make tight butt joints in conveyor belts with Flexco HD belt fasteners. It also illustrates step by step the latest practice in repairing rips and putting in patches.

26 DIAMOND CORE DRILLS—E. J. Longyear Co.—Core drilling is the most dependable method of proving up your mineral values. You will find the Pioneer Straitline well suited for this purpose. Bulletin 69 will tell you how to core drill your properties with Pioneer Straitlines.

27 HIGH-SPEED SYNCHRONOUS MOTORS — General Electric Co.—Your nearest G.E. representative will show you in detail the cost-cutting advantages of their new Tri-Clad high-speed synchronous motors. Now you can get, for your toughest ore concentrating and smelting jobs, triple-tough motors with proved extra protection features. For more data ask for Bulletin GEA-5113.

28 FIRST-AID MINE EQUIPMENT—Mine Safety Appliances Co.—Large and completely equipped miners' first-aid cabinet. All weather first-aid kits. Training and contest outfits. Folding stretcher outfits. Descriptive bulletins on request.

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31 FLOTATION INDEX — Dow Chemical Co.—To maintain the most complete, authoritative guide to published mercial on the flotation process, ask for a copy of the 19th addition to the Flotation Index.

32 TRUCO DIAMOND DRILLING PRODUCTS—Wheel Trueing Tool Co.—An interesting and informative new 32-page catalog of Truco Diamond Drilling Products will soon be available. Reserve your copy now. Diamond drill bits for mining one of the products described.

33 MARCY MILLS—Mine & Smelter Supply—On most ores, depending partly on hardness of ore and size of feed, it has been proven time after time that Marcy low pulp line grate discharge increases tonnage capacity 30% to 50%. Marcy Catalog 101 presents the evidence.

34 SURVEYING ALTIMETERS — American Paulin System—The newly improved surveying altimeter is described in a booklet "Origin and Development of the Altimeter." This instrument is being widely used by leading field engineers and geologists.

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Mijnbouwkundige

The above word is not a typesetter's error, but the Dutch term for mining, and it's a good selling point for a new book published in the Netherlands, called "Mining Nomenclature."

The 435 page volume contains, in five languages, most of the technical terms used in both mining and petroleum industries, arranged systematically, and then alphabetically, for easy reference. This knowledge of Dutch, French, German, Spanish and English can be provided by Tice & Lynch, 21 Pearl Street, New York, for about $5.50.

The Illinois Story

The proceedings of the 56th annual meeting of the Illinois Mining Institute have just been published, include six technical papers presented at the November sessions in Springfield, and ten reprints of important articles on mining technology published during 1948. The 234-page volume is well-peppered with a rich assortment of material on cleaning plants, dryers, power, roof supports, haulage, and methods of coal preparation, and everything is profusely illustrated. One hundred and fifty-six pages of pertinent advertising and a directory of Institute members complete this well-prepared record.

Reports on Idaho

Idaho's Bureau of Mines and Geology has issued two pamphlets, each describing the geology of a different section of their state. Pamphlet No. 82, "The Geology of Part of the South Slope of the St. Joe Mountains," presents geological information about some of the lesser known districts, including the Slate Creek, Black Prince, and the southern half of Placer Center mining districts of Shoshone County.

Pamphlet No. 83, entitled "Silver-Gold Deposits of the Yankee Fork District, Custer County," sheds light on the geology and deposits of this once-famous and highly productive district, about which the Bureau has had numerous requests for information.

The volumes are 75 cents apiece, from the University of Idaho, Moscow.

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Minerals for the Layman

Minerals and Mineral Deposits. By W. R. Jones and David Williams. Oxford University Press, 1949. 248 p. $2.

This book is an attempt to interest the general reader in many aspects of the subject of minerals, including their shape and internal structure, physical characters, mode of occurrence in rocks, and their importance in world affairs before and during the Industrial Age. It also tells something about the geological process of formation of economic mineral deposits, the search for new occurrences, the operations involved in the winning of industrial minerals and metals, and the distribution of mineral resources throughout the world.

Canadian Mineral Review

The Canadian Mineral Industry in 1946. Mines, Forests and Scientific Services Branch, Bureau of Mines, Ottawa. 1948. 137 p. 25¢.

A primary purpose of the annual reviews of the Canadian mineral industries is to provide specific or general information on domestic sources of mineral supply, trade in minerals and their products, uses, and developments in the industry for Canadians who have been showing a widening interest in matters concerned with the development of their mineral resources. The review is divided into metals, industrial minerals, and fuels, arranged in alphabetical order for easy reference.

California Underground

Minerals of California. Bull. 136, State of California, Dept. of Natural Resources, San Francisco. 1948. 402 p.

Herein are descriptions of 516 definite minerals besides many subspecies or varieties, about one-quarter of the total in the world, found in the state of California. California is famous for the discovery of new mineral species, 56 of which were originally found for the first time there; 39 of these 56 species have, to date, not been found elsewhere in the world. The minerals are arranged in alphabetical order, with a complete system of cross-referencing. Occurrences are listed by counties for each mineral and when of particular importance or interest are accompanied by a brief description of their geologic setting.

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